STUDY OF STABILITY OF OVERBURDEN DUMPS MIXED WITH FLYASH IN AN OPENCAST COAL MINE

A THESIS SUBMITTED IN PARTIAL FULFILLMENT OF THE REQUIREMENTS FOR THE DEGREE OF

BACHELOR OF TECHNOLOGY

IN

MINING ENGINEERING

BY

RAJ CHAKRAVARTY

109MN0106

DEPARTMENT OF MINING ENGINEERING

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ROURKELA – 769 008

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UNDER THE GUIDANCE OF

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CERTIFICATE

This is to certify that the thesis entitled, "***Study of Stability of Overburden Dumps Mixed with Flyash in an Opencast Coal Mine***" submitted by Sri **Raj Chakravarty, 109MN0106** in partial fulfillment for the award of Bachelor of Technology in Mining Engineering at National Institute of Technology Rourkela, is a record of original research work carried out under our supervision. The contents of this thesis have not been submitted elsewhere for the award of any degree what so ever to the best of our knowledge.

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CONTENTS

ABSTRACT

The huge quantities of fly ash generated in India (170 MT in 2012) by the coal fired thermal power plants every year raise serious question about their disposal besides creating adverse effects on the local environment. According to MoEF guidelines, any mine situated within 50 km from a power plant must use at least 25% flyash as its backfill material. In this project the stability of overburden dumps mixed with fly ash at KTK opencast mine of SCCL was studied. Samples of overburden and fly ash were collected from KTK mine and APGENCO, Chelpur respectively. Different geo-technical parameters i.e. density, cohesion and friction angle of OB mixed with 15% and 30% fly ash were determined through Standard Procter Hammer test and Direct Shear test.

Dumps of 30 m height were modelled in FLAC SLOPE to find out the safe slope angle i.e. angles for which the factor of safety > 1.2. From the present investigations the following conclusions were drawn:

Soil	**Angle (º)**
OB	29
OB + 15% fly ash	26
OB + 30% fly ash	28

- The initial decrease in slope angle from 29º to 26º with the addition of 15% flyash might be attributed to the inadequate packing of voids between OB particles by the finer sized flyash particles
- With increasing quantity of flyash i.e. at 30%, packing of the voids would become more compact as they reduce the void ratio. This would lead to the increase in slope angle obtained with OB + 30% fly ash from 26 º to 28º. However, there was no significant change in slope angle with addition of flyash vis-à-vis OB.

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CHAPTER 1

INTRODUCTION

INTRODUCTION

Coal has been the backbone of the Indian power sector. Indian coal typically is of low grade having an ash content of 40% in comparison to imported coals which have an ash content of 10-15%. Large quantities of ash (170 mn tonnes in 2012) [1] are generated by the thermal power stations in the country, which pollutes the environment. In addition to that, the availability of land for disposal of fly ash in slurry form in ash ponds is very difficult.

Keeping this in mind, the Minstry of Environment and Forests (MoEF) has issued notifications stipulating targets for 100% utilization of fly ash in a phased manner. For the mining industry it has directed the mines lying within 50 km of a thermal power plant (by road) to use at least 25% of the backfill material as flyash on a weight to weight basis subject to the approval of DGMS [2]. Proper scientific studies are necessary to evaluate the stability of such dumps.

Problems of slope instability occur frequently and are a source of major concern in the mining industry. These are caused either due to improper design of slopes or an incorrect assessment of the existing ones and pose a danger to the safety of people, equipment and other property. Geological structure, angle of the slope, weight acting on the slope, water content are some of the factors that affect slope stability and must be considered while analyzing the stability of a slope.

In this context the purpose of this project is to study the stability of overburden dumps mixed with fly ash at Kakatiya Khani Opencast (KTK OC) mine of Singareni Collieries Company Ltd. (SCCL) located in Bhupalpalli, Andhra Pradesh. The thermal power plant of APGENCO, situated around 15 km from the mine at Chelpur, supplies the fly ash.

1.1 Objectives of the Project

This project has the following objectives:

- To determine the geo-technical parameters of three different mixtures i.e. OB, OB+15% flyash and OB+30% flyash.
- To model the dump slopes in FLAC SLOPE to evaluate the factor of safety (FoS) for different slope angles.
- To propose safe slope angle for each of the different mixture of flyash and OB.

CHAPTER 2

LITERATURE REVIEW

LITERATURE REVIEW

2.1 Slope Stability

Slope stability, in general, indicates the resistance of a rock slope or dump slope to disintegration and subsequent flow. The ever increasing pit depths and production requirements from opencast mines subject the design engineers and planners to the pressure of working under the constraints of two conflicting requirements. On one hand economics could be improved by steepening the slope thereby reducing the amount of waste excavation. On the other hand higher slope angles mean a higher probability of failure of the slope leading to a loss of life, equipment and property This scenario poses a big question as to how to achieve an optimum design i.e. a compromise between economics and safety. The practical approach to slope stability is guided by various geo-technical parameters and a good measure of engineering judgment.

Judicious planning and implementation of an appropriate slope monitoring program can help in identifying the vulnerable slope sections, predict instabilities, evolve control strategies and even mining under unstable conditions.

Fig 2.1: Large Scale Slope Failure in Bingham Canyon Mine, Utah, 2013.

2.2 Factors Affecting the Stability of a Slope

- **Geometry of the slope:** The geometry of the slope is the most important factor which affects its stability. The basic geometrical slope design parameters are height of the bench, overall slope angle and the total area of failure surface. Stability of slope decreases with increases in height and slope angle. The curvature of the slope has profound effect on the instability and therefore convex section slopes should be avoided in the slope design. Greater the slope angle and higher the height less is the stability [3].

- **Geological Structure:** A rock slope may become unstable and fail along pre-existing structural discontinuities, by failure through intact material or by failure along a surface formed partly along discontinuity and partly through intact material. Instability may occur if the strata dips into the excavations. Localized steepening of strata is critical for the stability of the slopes. Stability is hampered if a clay band comes in between the two rock bands. Bedding planes and Joints are also zones of weaknesses.

  Stability of the slope is therefore dependent on the shear strength available along such surface, on their orientations with respect to the slope and water pressure action on the surface. These shear strength that is available along joint surface depends on the functional properties of the surface and the effective stress which are transmitted normal to the surface. Joints can create a situation where the failure planes involve a combination of joint sets that provide a cross over surface

  .

- **Lithology:** The rock materials constituting a pit slope determines the rock mass strength modified by discontinuities, folding, faulting, old workings and weathering. Low rock mass strength is characterized by quasi-circular raveling and rock fall instability like the formation of slope in massive sandstone restricts stability. Pit slopes containing soil alluvium or weathered rocks have low shearing strength and it is further reduced if water seeps through them. These types of slopes should be flatter.

- **Ground Water:** Excess water content in a slope reduces the cohesion and frictional parameters and also the normal effective stress. It causes increased up thrust and has adverse effect on the stability of the slopes. The chemical and physical effect of pure water pressure in joints filling material can thus alter the cohesion and friction of the discontinuity surface.

It provides uplift on the joint surfaces and reduces the frictional resistance. This in turn reduces the shearing resistance along the probable failure plane by reducing the effective normal stress on it. The effect of the water pressure in the rock pores causes a decrease in the compressive strength predominantly where the confining stress has been reduced.

- **Mining Method:** Essentially there are four methods of advance in surface mining. They are:
    - Strike cut- advancing down the dip
    - Strike cut- advancing up the dip
    - Dip cut- along the strike
    - Open pit working

  The use of dip cuts with advance on the strike reduces the time and length that a face is exposed during excavation. Dip cuts which advance in an oblique manner to strike are used to reduce the strata dip in to the excavation. The Open pit method is used in sharply dipping seams because the greater slope height is more prone to buckling modes of failure. Dip cut is the most stable method of working but it suffers from restricted production potential. In circular failures spoil dumps are more common. Mining equipment which piles on the benches of the open pit mine gives rise to the increase in surcharge, which in turn increases the downward pulling force on the slope and thus instability occurs.

- **Time:** The time for which a slope has to stand after excavation should be considered as well. The slopes that are generally found in surface mines have to stand for a short time but they encounter the same rigorous treatment as in civil projects. In the long term, the progressive strain softening rate is a significant factor in the slope stability.

- **Dynamic Forces:** Vibrations due to blasting momentarily increases the shear stress as a result dynamic acceleration of material and thus increases the stability problem in the slope face. Blasting is a crucial factor in deciding the maximum attainable bench face angles. The effects poor blasting can be significant for bench stability [4]. In addition to blast damage and back break both of which reduce the bench face angle, blasting vibrations could potentially cause failure of the rock mass. For small slopes, smooth blasting techniques have been proposed and the experiences are quite good. For large slopes, blasting is less of a problem because back break and blast damage have minor effects on the overall slope angle.

Moreover, the high frequencies of the blast acceleration waves exclude them from displacing large rock masses uniformly [4]. Seismic events, i.e., low frequency vibrations, could be more precarious for large scale slopes and several failures of natural slopes have been witnessed in mountainous areas. External loading also plays an important role as in case of surcharge due to dumps on the crest of the benches.

- **Cohesion:** The resistance force per unit area is termed as cohesion, and is measured in Pascal (Pa). In natural soils, cohesion arises from electrostatic bonds between clay and silt particles. Thus, soils empty of clay or silt are not cohesive but for capillary forces arising when little water forms bridges between sand grains, causing negative pore pressure (or "suction"). Values of soil cohesion usually are of the order of some kPa. Rocks typically display much greater cohesion, thousands of times higher than soils. At finite normal stresses, soils and rocks normally display both cohesive and frictional behavior. The shear strength of a soil is thus the sum of the cohesive and frictional contributions. Higher is the cohesion value, more stable will be the slope [5].

- **Angle of Internal Friction:** It is the measure of the angle between the normal force and resultant force when failure just occurs in reaction to a shearing stress. It is an indicator of the ability of a rock or soil to withstand shear stress. Angle of internal friction is depends upon particle roundness and particle size. Lower roundness or larger median particle size results in larger friction angle. The sands with less quartz contained greater amounts of potassium-feldspar, plagioclase, calcite, and/or dolomite and these minerals generally have higher sliding frictional resistance compared to that of quartz. Angle of internal friction, can be determined in the laboratory by the Direct Shear Test or the Triaxial Shear Test.

- **Old workings:** Old workings affect the stability of a slope in numerous ways. They can act as channels for groundwater flow, many of them might be unstable and collapse when subjected to weights.

2.3 Types of Slope Failure

2.3.1 Plane Failure

A plane failure is a comparatively rare sight in rock slopes because it is only occasionally that all the geometric conditions required to produce such a failure occur in an actual slope. For a plane failure to occur, the plane on which sliding occurs must strike parallel or nearly parallel (within approximately ±20°) to the slope face and the sliding plane must "daylight" in the slope face, which means that the dip of the plane must be lower than that of the slope face. It is a rare sight because the geometric conditions interact in a much more complex manner in reality. However it is very useful to demonstrate the sensitivity of slope to changes in shear strength or ground water conditions.

2.3.2 Wedge Failure

Wedge failures result when rock masses slide along two intersecting discontinuities both of which dip out of the cut slope at an oblique angle to the cut face, forming a wedge-shaped block. Commonly, these rock wedges are exposed by excavations that daylight the line of intersection that forms the axis of sliding, precipitating movement of the rock mass either along both planes simultaneously or along the steeper of the two planes in the direction of maximum dip. Depending upon the ratio between peak and residual shear strengths, wedge failures can occur rapidly, within seconds or minutes, or over a much longer time frame, or on the order of several months. The size of a wedge failure can range from a few cubic meters to very large slides from which the potential for destruction can be enormous [8]. Rock masses with well-defined orthogonal joint sets or cleavages in additional to inclined bedding or foliation generally are favorable situations for wedge failure. Shale, thin-bedded siltstones, claystones, limestones, and tend to be more prone to wedge failure development than other rock types.

2.3.3 Circular Failure

in the case of a closely fractured or highly weathered rock, a strongly defined structural pattern no longer exists, and the slide surface is free to find the line of least resistance through the slope. Observations of slope failures in these materials suggest that this slide surface generally takes the form of a circle, and most stability theories are based upon this observation. The conditions under which circular failure will occur arise when the individual particles in a soil or rock mass are very small compared with the size of the slope. Hence, broken rock in a fill will tend to behave as a "soil" and fail in a circular mode when the slope dimensions are substantially greater than the dimensions of the rock fragments. Similarly, soil consisting of sand, silt and smaller particle sizes will exhibit circular slide surfaces, even in slopes only a few meters in height [9].

2.3.4 Toppling Failure

Toppling failures most commonly occur in rock masses that are subdivided into a series of columns formed by a set of fractures that strike approximately parallel to the slope face and dip steeply into the face. In a toppling failure the rock column or slab rotates about a fixed point at or near the base of the slope at the same time that slippage occurs between the layers. Rock types most susceptible to this mode of failure are columnar basalts and sedimentary and metamorphic rocks with well-defined bedding planes. There are several types of toppling failures, including flexural, block, or a combination of block and flexural toppling. Toppling can also occur as a secondary failure mode associated with other failure mechanisms such as block sliding [10].

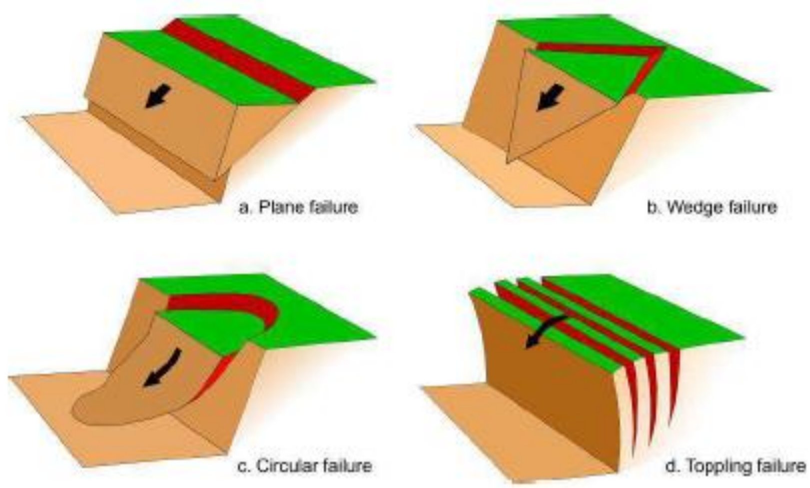

Fig 2.2 Four Different Types of Slope Failure Modes.

2.4 Slope Stability Analysis

2.4.1 Limit Equilibrium Method

The stability of rock slopes for the geological conditions depends on the shear strength generated along the sliding surface. For all shear type failures, the rock can be assumed to be a Mohr–Coulomb material in which the shear strength is expressed in terms of the cohesion c and friction angle φ [6].

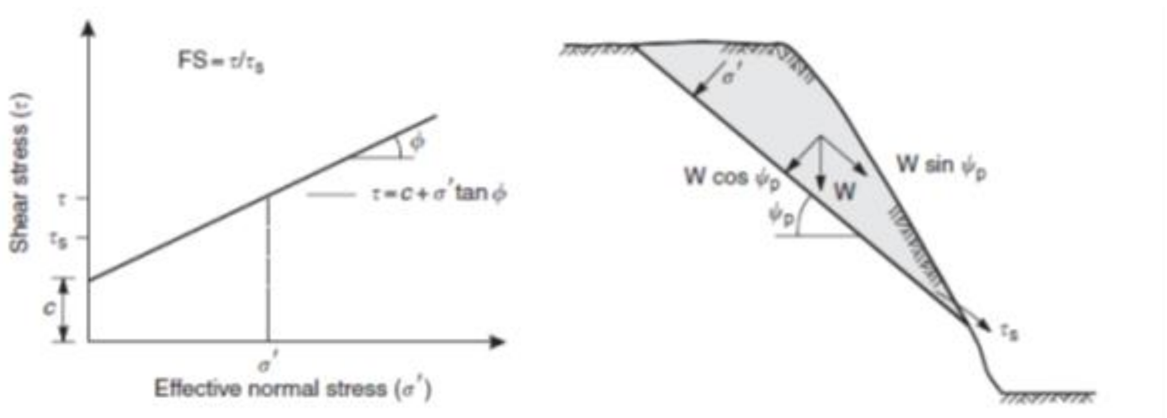

Fig 2.3: Mohr Diagram Showing Shear Strength Defined by Cohesion c and Friction Angle φ and the Resolution of Weight W.

For a sliding surface on which there is an effective normal stress σ acting, the shear strength τ developed on this surface is given by $\tau = c + \sigma \tan \varphi$. (1)

Calculation of the factor of safety for the block shown in Figure 2.3 involves the resolution of the force acting on the sliding surface into components acting perpendicular and parallel to this surface. That is, if the dip of the sliding surface is ψ_p, its area is A, and the weight of the block lying above the sliding surface is W, then the normal and shear stresses on the sliding plane are

Normal Stress, $\sigma = (W \cos \psi_p / A)$ and Shear Stress, $\tau_s = (W \sin \psi_p / A)$ (2)

Now equation 1 becomes $\tau = c + (W \cos \psi_p \tan \varphi / A)$ (3)

The term W $\sin\psi_p$ defines the resultant force acting down the sliding plane and is termed the "driving force" (τsA), while the term [cA + W $\cos\psi$p $\tan\varphi$] defines the shear strength forces acting up the plane that resist sliding and are termed the "resisting forces" (τA).

The stability of the block can be quantified by the ratio of the resisting and driving forces, which is termed the factor of safety, FS. Therefore, the expression for the factor of safety is

FS = Resisting Forces / Driving Forces

$$= [cA + W \cos\psi_p \tan\varphi] / [W \sin\psi_p] \qquad (4)$$

If a surface is clean and dry then the cohesion will nearly be zero. Then in equation (4), FS = 1 if $\psi_p = \varphi$. The block of rock will slide when the dip angle of the sliding surface equals the friction angle of this surface, and that stability is independent of the size of the sliding block. That is, the block is at a condition of "limiting equilibrium" when the driving forces are exactly equal to the resisting forces and the factor of safety is equal to 1.0. Therefore, the method of slope stability analysis described in this section is termed limit equilibrium analysis.

2.4.2 Sensitivity Analysis

The factor of safety analysis described in the limit equilibrium method involves selection of a single value for each of the parameters that define the driving and resisting forces in the slope. In reality, each parameter has a range of values, and a method of examining the effect of this variability on the factor of safety is to carry out sensitivity analyses using upper and lower bound values for those parameters considered critical to design. However, to carry out sensitivity analyses for more than three parameters is cumbersome, and it is difficult to examine the relationship between each of the parameters. Consequently, the usual design procedure involves a combination of analysis and judgment in assessing the influence on stability of variability in the design parameters, and then selecting an appropriate factor of safety [6].

The value of sensitivity analysis is to assess which parameters have the greatest influence on stability. This information can be used to collect data that will define this parameter(s) more precisely. Alternatively, if there is uncertainty in the value of an important design parameter, this can be accounted for in design by using an appropriate factor of safety.

2.4.3 Probabilistic Design Methods

Probabilistic design is a systematic procedure for examining the effect of the variability of each parameter on slope stability. A probability distribution of the factor of safety is calculated, from which the probability of failure (PF) of the slope is determined. Probability analysis was first developed in the 1940s and is used in the structural and aeronautical engineering fields to examine the reliability of complex systems. Among its early uses in geotechnical engineering was in open pit mine slope design where a certain risk of failure is acceptable, and this type of analysis could be readily incorporated into the economic planning of the mine. The use of probability analysis in design requires that there be generally accepted ranges of probability of failure for different types of structure, as there are for factors of safety**.** For example, for open pit mine slopes for which slope performance is closely managed and there is little risk to life in the event of a failure, the acceptable range of annual probability of failure can be about 10^{-1} to 10^{-2}. In comparison, for dams where failure could result in the loss of several hundred lives, annual probability of failure should not exceed 10^{-4} to 10^{-5} [6].

2.5 Guidelines for Design of Dump Slopes

2.5.1 CMR Guidelines

Section 98 of The Coal Mine Regulations (CMR), 1957 stipulates that:

In alluvial soil, morum, gravel, clay, debris or other similar ground:

- the sides shall be sloped at an angle of safety not exceeding 45 degrees from the horizontal or such other angle as permitted by Regional Inspector of mines
- the sides shall be kept benched and the height of any bench shall not exceed 1.5 m and the breadth thereof shall not be less than the height
- In coal, the sides shall either be kept sloped at an angle of safety not exceeding 45 degree from the horizontal, or the sides shall be kept benched and the height of any bench shall not exceed 3m and the width thereof shall not be less than the height.
- In an excavation in any hard and compact ground or in prospecting trenches or pits, the sides shall be adequately benched, sloped or secured so as to prevent danger from fall of sides.

- No person shall undercut any face or side or cause or permit such undercutting as to cause any overhanging.

2.5.2 DGMS Guidelines

The Directorate General of Mine Safety (DGMS), Dhanbad is the regulatory body for enforcing safety aspects in the mining industry and it has issued the following guidelines regarding slope stability:

- Before starting a mechanized opencast working, design of the pit, including method of working and ultimate pit slope shall be planned and designed as determined by a scientific study.
- The height of the benches in overburden consisting of alluvium or other soft soil shall not exceed 5 m and the width thereof shall not be less than three times the height of the bench
- The height of the benches in overburden of other rock formation shall not be more than the designed reach of the excavation machine in use for digging, excavation or removal.
- The width of any bench shall not be less than
  - the width of the widest machine plying on the bench plus 2m,
  - if dumpers ply on the bench, three times the width of the dumper, or
  - the height of the bench, whichever is more.
- While removing overburden, the top soil shall be stacked at a separate place, so that, the same is used to cover the reclaimed area.
- The slope of a spoil bank shall be determined by the natural angle of repose of the material being deposited, but shall in no case exceed 37.5 degrees from the horizontal. The spoil bank shall not be retained by artificial means at an angle in excess of natural angle of repose or 37.5 degrees whichever is less.
- Loose overburden and other such material from opencast workings or other rejects from washeries or from other source shall be dumped in such a manner that there is no possibility of dumped material sliding.
- Any spoil bank exceeding 30m in height shall be benched so that no bench exceeds 30m in height and the overall slope shall not exceed 1 vertical to 1.5 horizontal.

- The toe of a spoil-bank shall not be extended to any point within 45m of a mine opening, railway or other public works, public road or building or other permanent structure not belonging to the owner.

2.6 Review of Research Work of Other Investigators

Chaulya et al. (1999): Maximum displacement of elements occurs near the crest of the dump. any dump deformation monitoring programme should be planned near the crest of the dump slopes as dump failure generally occurs after significant movement over a long time. Revegetation is one of the widely used technique for stabilisation of dump slopes. Stability investigation of a dump slope of Mudidih mine in Dhanbad revealed the cohesion and friction angle values of the 30 m high, 35.5° angle dump as 0.6 kg/cm^2 and 31.5º. The plantation of grasses enhanced the FoS of the dump from 1.2 to 1.4 for the same geometry of dump.

Singh et al. (2004): At Lajkura Opencast coal mine of MCL, 22m of overburden immediately above the Lajkura coal seam (having a thickness of 18 m) was being removed by a dragline. The dump material is mainly characterized by sandstone, shale and coal. Bulk density and direct shear tests were conducted on the samples collected from the mine. Kinematic analysis was included to determine the critical orientation of structural discontinuities. Detailed slope stability analysis was carried out using the GALENA software. Based on the analysis, the 40m high dragline highwall was likely to be safe with a 70° slope angle.

Jhanwar (2008): At the New Majri Opencast coal mine of WCL, a 150 m long failure occurred in the strike direction of the 60 m high pit slope. Geo-technical studies carried out indicated the cohesion and angle of internal friction values for the soil were 48 kPa and 23° respectively. The investigation also revealed the ingress of rainwater into the slope which would have increased the pore pressure and eventually led to the failure. Based on the analysis, overall slope angles of 25º and 28º were proposed for slope heights of 30 m and 20 m respectively.

Kainthola et al. (2011): Stability of waste dumps is very crucial due to the non-availability of land, weak rock conditions as well as heavy rains. The failed dump in a coal mine of Western Coalfields Ltd. had a height of 75 m with 43° slope angle which had slipped forward by 18m. Representative loose dump material samples were collected from the site and tested to determine the physico-mechanical properties of dump material. The slope was numerically modeled and based on back analysis; the above condition gave a FoS of 0.8. Owing to the weak geomechanical strength of the dump it was suggested to keep the flatter slope of 25° with a height of 75 m which had a higher FoS of 1.3.

Jayanthu et al. (2012), evaluated the stability of OB dumps mixed with 25 % flyash at the coal mine of Jindal Power Ltd, Raigarh. Density, Cohesion and Friction angle values for OB were 1870 kg/m^3, 41.79kN/m^2 and 28.5° respectively. The same values for OB with 25% fly ash were 1740 kg/m^3, 89.61 kN/m^2 and 22.92°. The dumps were modeled in PLAXIS software package. The total dump height was taken to be 120 m, divided into four decks of 30 m with a deck angle of 32° and an overall slope angle of 22°. The factor of safety for this model was found to be 1.75. As part of the reclamation policy the dump was also modeled with 2m of top soil (cohesion and friction angle of 78.2 kN/m^2 and 20.5°) at the surface. The second trial gave a factor of safety of 1.78 which suggests that the dump along with the top soil layer had improved stability.

CHAPTER 3

PROJECT METHODOLOGY

PROJECT METHODOLOGY

The work procedure of this project has been shown in the following flow sheet;

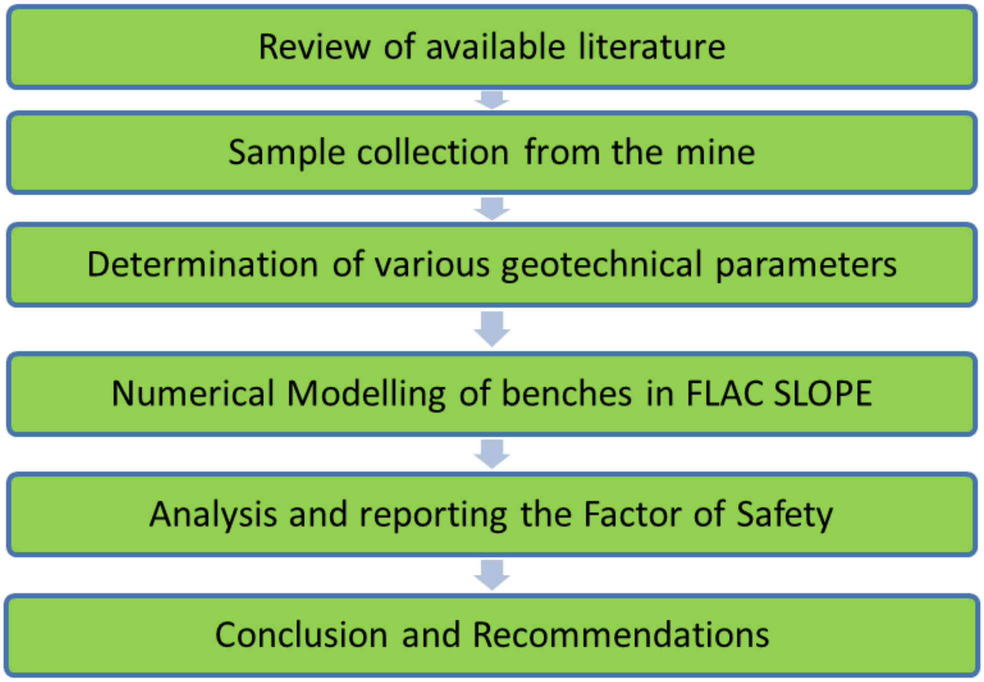

Fig 3.1: Flowsheet of the Project Methodology.

3.1 Description of the Study Area

KTK OC mine, SCCL is located at a distance of 3 km from Bhupalapalli. It has a leasehold area of 336 ha. The maximum depth of workings in the mine is upto 85 m. A total of 5 seams are being worked in the mine. Production started in 2009 with an annual target of 1.2 million tonnes of coal 13 million m^3 of OB removal. The stripping ratio of the mine ranges from 1:10 to 1:12. KTK OC mine is presently producing 50000 tonnes of coal per month. The total production is interlinked and is being transported to Kakatiya Thermal Power Station (KTPP) Chelpur located on the Bhupalapalli – Warangal PWD road at a distance of 15 km from the mine. KTPP Chelpur, a 500 MW power station, is presently producing 2200 TPD of fly ash and 600 TPD of Bottom ash. This is likely to be doubled with the installation of an additional 600 MW power plant now under construction.

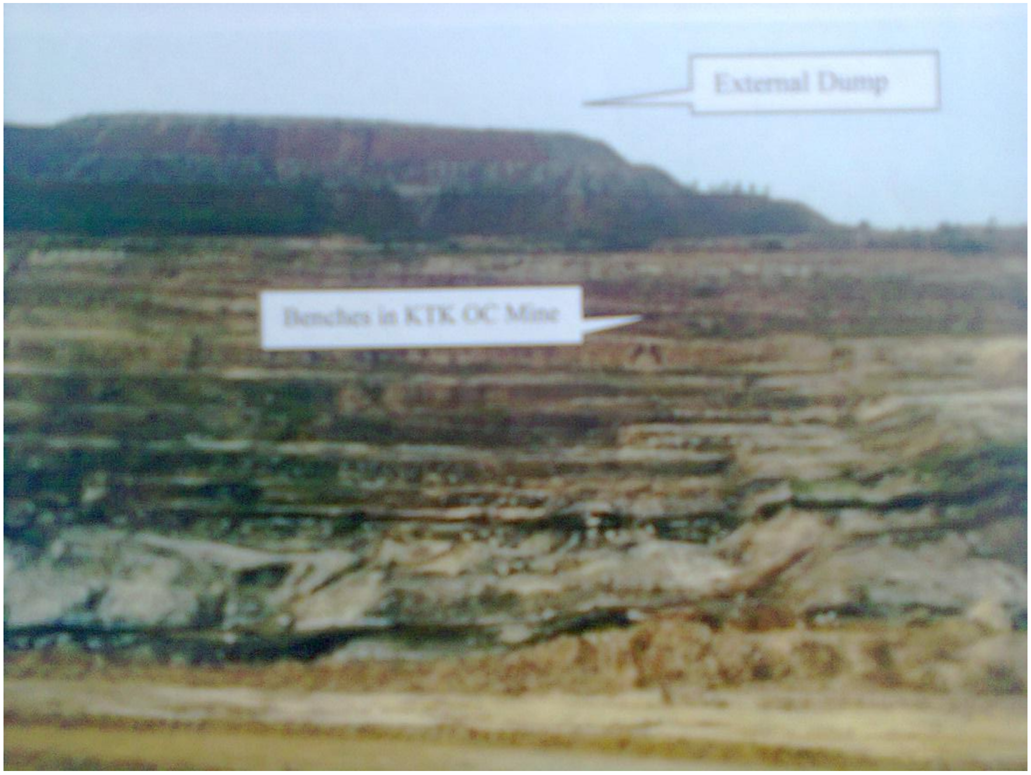

Fig 3.2: Existing Benches in the Mine and the External Dump

3.1.1 Method of Dumping Flyash and OB

The current method of transporting OB being practiced in the mine, i.e. hauling and dumping through dumpers followed by dozing, would be continued. The fly ash would be dumped in between the OB heaps at the rate of 30% of the OB material (approximately 3 trucks of OB and 1 truck of flyash). The same procedure will be followed for dumping 15% fly ash. While dozing the OB and fly ash heaps, a nearly homogenous mixture would be formed along the slope and it would progresses up to the boundary of the dump area. The existing practice of dumping OB is shown in Fig 3.2

Fig 3.3: Current Practice of Dumping OB

3.2 Sample Collection

OB samples were collected from the OB dump yard of the mine shown in figure 3.5. A cylindrical mould of length 15 cm and 10 cm internal diameter was hammered into the dump surface. The sides of the mould were cleared and it was carefully taken out. The contents were immediately placed in a gunny bag to avoid the effects of moisture on the sample. Flyash samples were collected from the ash pond (Fig 3.4) of KTPP, Chelpur.

Fig 3.4: Ash Pond of KTPP, Chelpur

Fig 3.5: KTK OC MINE

3.3 Experimental Analysis

The following geo-technical parameters are required to evaluate the factor of safety of a slope in FLAC SLOPE:

- Density (kg/m^3)
- Cohesion (Pa)
- Angle of Internal Friction (°)

In addition to these the grain size of the sample should be determined to characterize the type of soil. Therefore the following tests were conducted on OB, OB+15% fly ash, OB + 30% fly ash:

- Grain size analysis
- Procter hammer test – to determine density
- Direct Shear test – to determine 'c' and 'φ'.

3.3.1 Grain Size analysis [14]

Soil is a porous mass consisting of aggregates of particles of different shapes and sizes that are held together by inter-particulate electrochemical forces. Thus the variations in size of particles of the grains in a soil mass can form one of the basis of classification of soils. Though grain size particle distribution in soil is not adequate to predict engineering properties of soils, it provides enough information to classify the soil as coarse grained or fine grained. Soil fraction with size greater than 0.075 mm is known as coarse and lesser than that as fines.

Table 3.1: Different Fractions of Soil According to the Particle Size

Particle Size	Fraction
> 4.75 mm	Gravel
0.075 mm – 4.75 mm	Sand
0.002 mm – 0.075 mm	Silts
< 0.002 mm	Clay

The sieves were arranged on top of one another such that the coarsest one was at the top and the finest one at the bottom. 1 kg of oven dried soil sample was taken and placed on the coarsest sieve. The entire assembly of sieves was placed on the sieve shaker and shaken for about 10 min. The material retained on each sieve was recorded in a tabular format and the cumulative percentage retained was calculated. The cumulative percentage of fines was also calculated and the graph between percentage of fines and grain size was plotted.

The following observations were obtained for the different samples

Sample: OB

Amount of sample taken: 1000 gm

Table 3.2: Grain Size Analysis of OB Sample

Sieve Size (mm)	Weight Retained (gm)	Cumulative weight (gm)	% age weight retained	%age finer
4.75	52.6	52.6	5.26	94.74
2	63.6	116.2	6.36	93.64
1	223.1	339.3	22.31	77.69
0.425	249.3	588.6	24.93	75.07
0.212	179.8	768.4	17.98	82.02
0.15	27.8	796.2	2.78	97.22
0.075	21.6	817.8	2.16	97.84
0.01	180.6	180.6	18.06	0

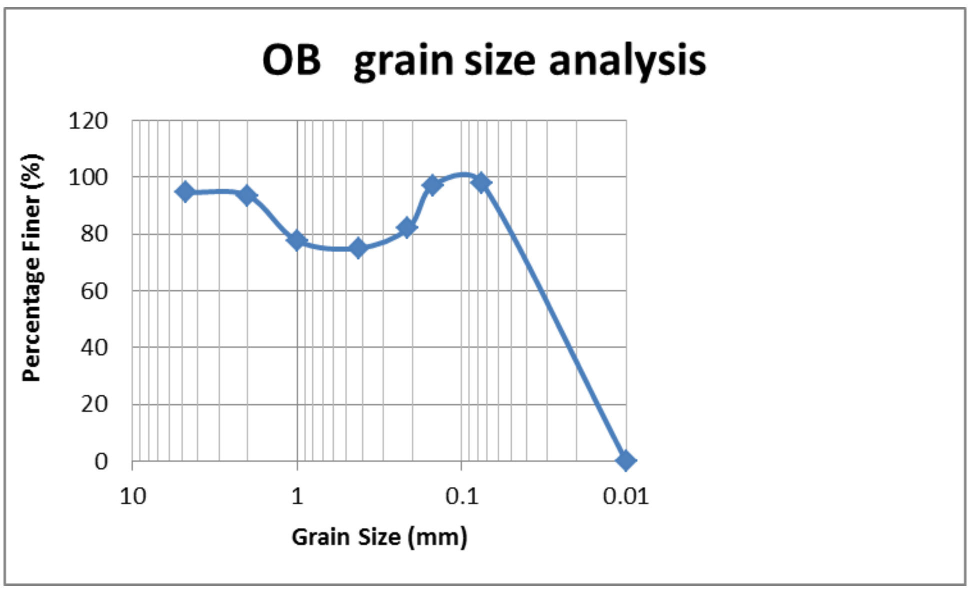

Fig 3.6: Grain Size Curve of OB Sample

Sample: OB + 15% flyash

Amount of sample taken: 998 gm (848 gm OB + 150 gm flyash)

Table 3.3: Grain Size Analysis of OB + 15% Flyash Sample

Sieve Size (mm)	Weight Retained (gm)	Cumulative weight (gm)	% age weight retained	%age finer
4.75	32	32	3.206	96.79
2	61	93	9.319	90.68
1	186	279	27.956	72.04
0.425	328	607	60.822	39.18
0.212	318	925	92.685	7.31
0.15	45	970	97.194	2.81
0.075	11	981	98.297	1.70
0.01	4	985	98.697	1.30

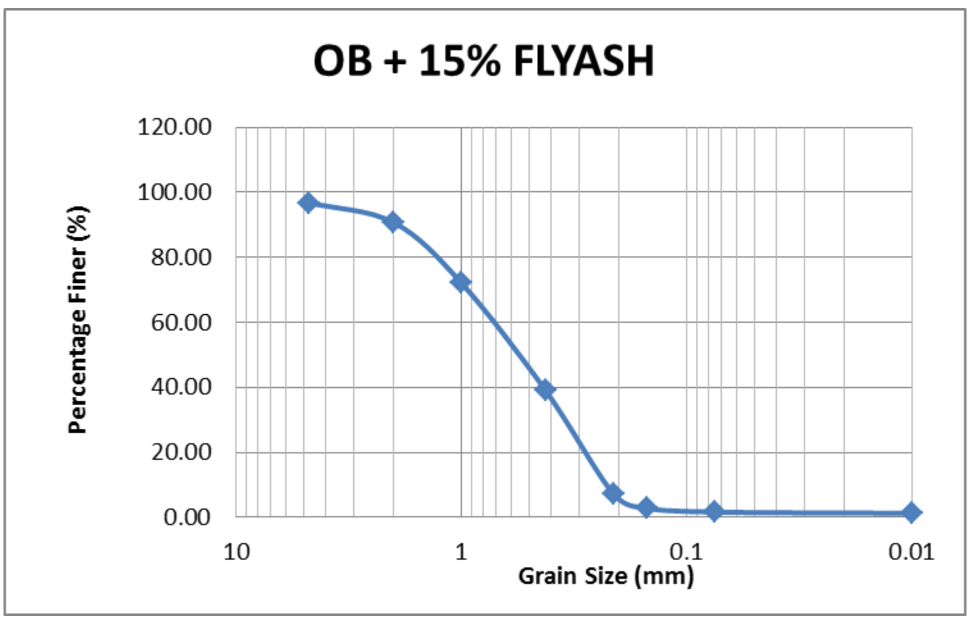

Fig 3.7: Grain Size Curve of OB + 15 % Flyash Sample

Sample: OB + 30% fly ash

Amount of sample taken: 1000 gm (700 gm OB + 300 gm flyash)

Table 3.4: Grain Size Analysis of OB + 30% Flyash Sample

Sieve Size (mm)	Weight Retained (gm)	Cumulative weight (gm)	% age weight retained	%age finer
4.75	43	43	4.30	95.70
2	76	119	11.90	88.10
1	222	341	34.10	65.90
0.425	338	679	67.90	32.10
0.212	193	872	87.20	12.80
0.15	61	933	93.30	6.70
0.075	41	974	97.40	2.60
0.01	19	993	99.30	0.70

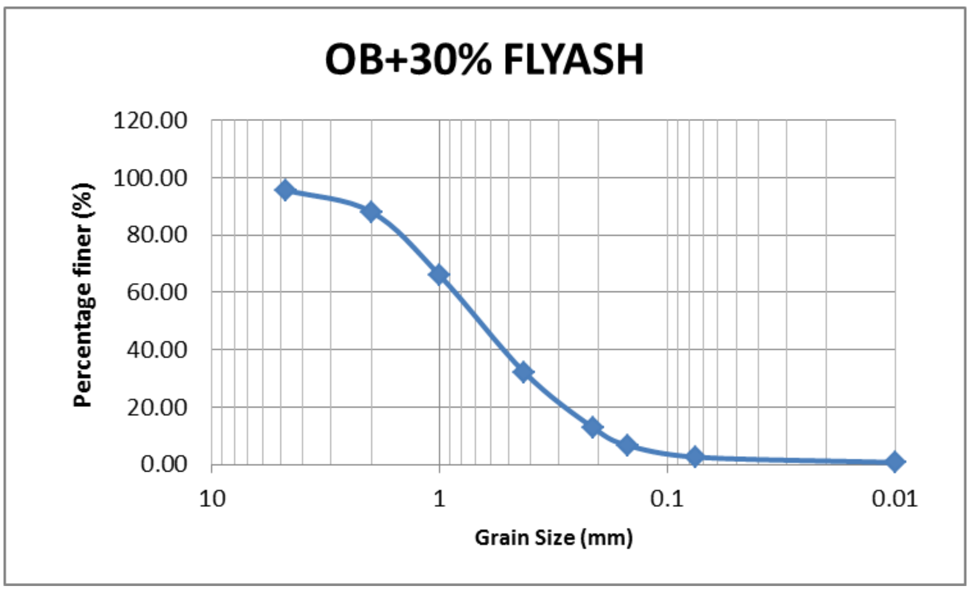

Fig. 3.8: Grain Size Curve of OB + 30% Flyash Sample

From the above observations it is clear that samples are sandy in nature.

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3.3.2 Standard Procter Hammer Test [13]

This test determines the optimum amount of water to be mixed with a soil in orer to obtain maximum compaction for a given compactive effort. Maximum compaction leads to maximum dry density and hence the deformation and strength characteristics of soils turn out to be the best possible value.

This test is satisfactory for cohesive soils but does not lend itself well to the study of compaction characteristics of clean sands and gravel which are easily displaced when compacted with rammer. When high densities are warranted as in the case of formation for airport runways compactive effort becomes necessary. For this a modified procter test is adopted.

Procedure:

The empty mould was weighed (W_m) and fixed to the base plate. Thereafter the collar was attached. 2.5 kg of sample was taken and 100 ml of water added to it. It was then thoroughly mixed. The wet sample was divided into roughly three parts. The mould was filled with one part of the soil and compacted with 25 evenly distributed blows with the standard rammer. The next part was then added to the mould and the blows were repeated. This step was continued till all the parts are had been compacted. The collar was then removed and the top of the soil was trimmed to fit within the mould. The mould was detached from the base plate and its weight was recorded. Some amount of soil from the mould was taken in a tin container to determine the moisture content. The soil wass then added with 50 ml water and above steps were repeated.

Fig. 3.9: Compaction of Soil in the Mould with the Rammer

The following observations were obtained from the procter hammer test:

Sample: OB

Table 3.5: Procter Compaction Test for OB

WEIGHT OF SAMPLE,	**W_m** = 2.5 kg
WEIGHT OF EMPTY MOULD,	**W_E** = 1.902 kg
INTERNAL DIAMETER OF MOULD,	**d** = 10 cm
HEIGHT OF MOULD,	**h** = 12.7 cm
VOLUME OF MOULD,	**V** = 997.45 cc

PARAMETER		1	2	3	4	5
WEIGHT OF MOULD + SOIL,	**W_1** (gm)	3790	3914	4038	4112	4088
WEIGHT OF COMPACTED SOIL,	**W_c** (gm)	1888	2012	2136	2210	2186
WET DENSITY,	**d_w** = W_c/V (g/cc)	1.892	2.017	2.141	2.215	2.191
WEIGHT OF CONTAINER,	**X_1** (gm)	19.97	20.99	19.33	21.53	21.59
WEIGHT OF CONTAINER + WET SOIL,	**X_2** (gm)	116.8	72.6	118.7	125.2	111.4
WEIGHT OF CONTAINER + DRY SOIL,	**X_3** (gm)	112.7	70.1	111.4	116.5	102.4
WEIGHT OF DRY SOIL,	$X_3 - X_1$ (gm)	92.73	49.11	92.07	94.97	80.81
WATER	$X_2 - X_3$ (gm)	4.1	2.5	7.3	8.7	9
WATER CONTENT,	**W** = $(X_2 - X_3)/(X_3 - X_1)$ (%)	4.42	5.09	7.93	9.16	11.14
DRY DENSITY,	**d_d** = $d_w/(1 + 0.01W)$ (g/cc)	1.812	1.919	1.984	2.029	1.971

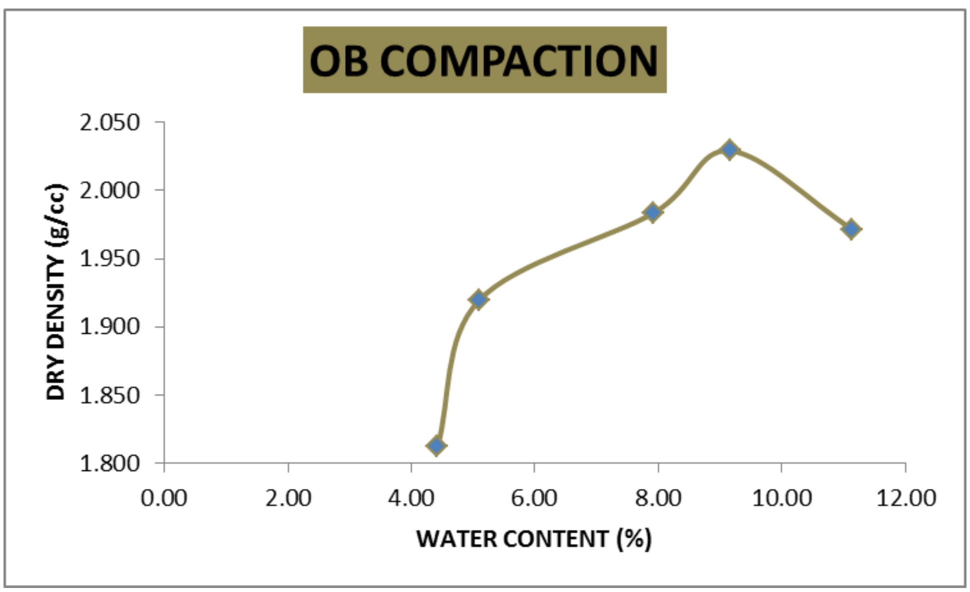

Fig. 3.10: Compaction Curve for OB

Sample: OB + 15% flyash

Table 3.6: Procter Compaction Test for OB + 15% Flyash

WEIGHT OF SAMPLE, **W_m** = 2.5 kg
WEIGHT OF EMPTY MOULD, **W_E** = 1.884 kg
INTERNAL DIAMETER OF MOULD, **d** = 10 cm
HEIGHT OF MOULD, **h** = 12.5 cm
VOLUME OF MOULD, **V** = 981.74 cc

PARAMETER	1	2	3	4	5
WEIGHT OF MOULD + SOIL, **W_1** (gm)	3702	3810	3934	3964	3964
WEIGHT OF COMPACTED SOIL, **W_c** (gm)	1818	1926	2050	2080	2080
WET DENSITY, **d_w** = W_c/V (g/cc)	1.852	1.962	2.088	2.119	2.119
WEIGHT OF CONTAINER, **X_1** (gm)	19.29	19.97	20.88	21.21	21.53
WEIGHT OF CONTAINER + WET SOIL, **X_2** (gm)	94.90	100.80	91.40	105.30	94.60
WEIGHT OF CONTAINER + DRY SOIL, **X_3** (gm)	91.10	95.40	85.40	96.80	82.60
WEIGHT OF DRY SOIL, $X_3 - X_1$ (gm)	71.81	75.43	64.52	75.59	61.07
WATER $X_2 - X_3$ (gm)	3.80	5.40	6.00	8.50	12.00
WATER CONTENT, **W** = $(X_2 - X_3)/(X_3 - X_1)$ (%)	5.29	7.16	9.30	11.24	19.65
DRY DENSITY, **d_d** = $d_w/(1 + 0.01W)$ (g/cc)	1.759	1.831	1.910	1.905	1.771

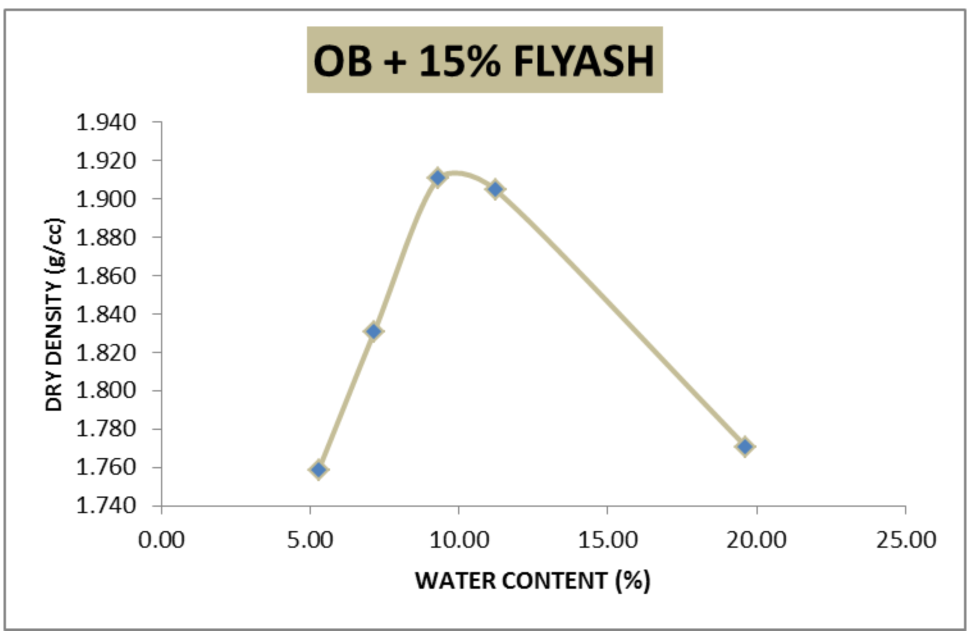

Fig. 3.11: Compaction Curve for OB + 15% Flyash

Sample: OB + 30% fly ash

Table 3.7: Procter Compaction Test for OB + 30% Flyash

WEIGHT OF SAMPLE, W_m = 2.5 kg							
WEIGHT OF EMPTY MOULD, W_E = 1.910 kg							
INTERNAL DIAMETER OF MOULD, **d** = 10 cm							
HEIGHT OF MOULD, **h** = 12.6 cm							
VOLUME OF MOULD, **V** = 989.60 cc							
PARAMETER	**1**	**2**	**3**	**4**	**5**	**6**	**7**
WEIGHT OF MOULD + SOIL, W_1 (gm)	3484	3560	3622	3686	3760	3860	3856
WEIGHT OF COMPACTED SOIL, W_c (gm)	1574	1650	1712	1776	1850	1950	1946
WET DENSITY, $d_w = W_c/V$ (g/cc)	1.591	1.667	1.730	1.795	1.869	1.970	1.966
WEIGHT OF CONTAINER, X_1 (gm)	19.29	19.97	20.88	21.21	21.53	19.76	19.68
WEIGHT OF CONTAINER + WET SOIL, X_2 (gm)	87.80	93.90	97.40	97.00	93.90	103.18	95.40
WEIGHT OF CONTAINER + DRY SOIL, X_3 (gm)	84.00	88.80	90.60	89.10	85.40	91.70	84.10
WEIGHT OF DRY SOIL, $X_3 - X_1$ (gm)	64.71	68.83	69.72	67.89	63.87	71.94	64.42
WATER $X_2 - X_3$ (gm)	3.80	5.10	6.80	7.90	8.50	11.48	11.30
WATER CONTENT, $W = (X_2 - X_3)/(X_3 - X_1)$ (%)	5.872	7.410	9.753	11.636	13.308	15.958	17.541
DRY DENSITY, $d_d = d_w/(1+0.01W)$ (g/cc)	1.50232	1.55232	1.576255	1.607597	1.649873	1.69932	1.67299

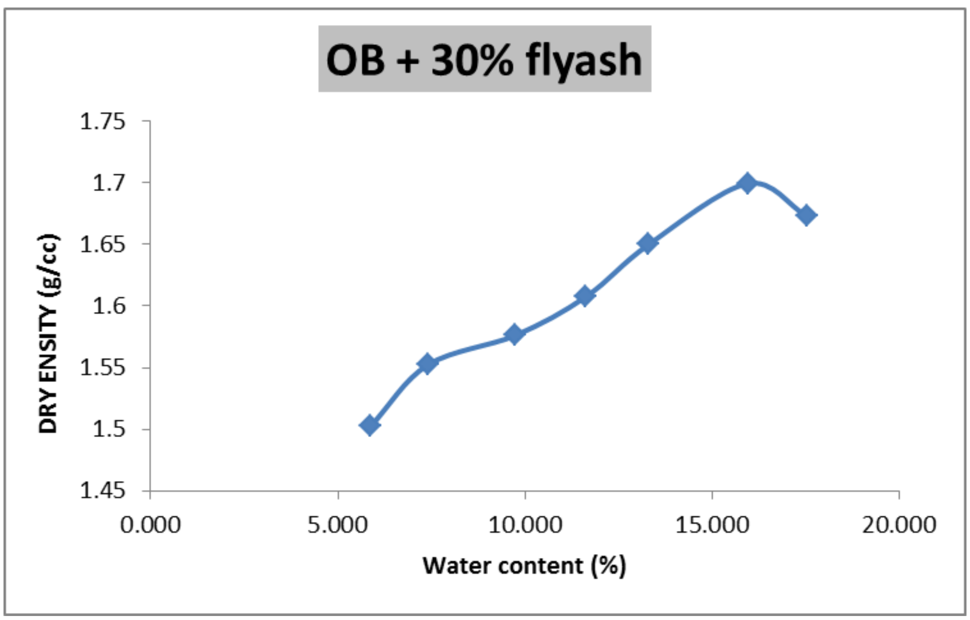

Fig. 3.12: Compaction Curve for OB + 30% fly ash

The maximum of the compaction curve gives denotes the maximum dry density and the corresponding optimum moisture content. Following were the results of the procter compaction test;

Table 3.8: Results of Procter Compaction Test

Sample	**MDD (g/cm^3)**	**OMC (%)**
OB	2.02	9.16
OB + 15% fly ash	1.91	10.11
OB + 30% fly ash	1.70	15.95

3.3.3 Direct Shear Test [12]

Shear strength in a soil is derived from the surface frictional resistance along the sliding plane , interlocking between individual rock grains and cohesion in sliding surface of soil model. The shear strength of soil is given by Mohr-Coulomb expression: $\boldsymbol{\tau = c + \sigma \tan \varphi}$ where ‘τ’ = Shear Strength, ‘σ’ = Normal Strength in failure plane, ‘c’ = cohesion, φ = angle of internal friction.In a test of soil, there are two basic stages. First nominal load is applied to specimen and then failure is induced by applying a shear stress. If no water is allowed to escape from or enter into specimen either during consolidation is undrained test. If the specimen is allowed to consolidate under normal load but no drainage of water is allowed during shear, it is called consolidated undrained or consolidated quick test.

The dimensions of the shear box were measured and the mass of the sample to be tested was determined. The required mass of sample was taken in a tray and water added to it at its optimum moisture content. It was then mixed thoroughly. The shear box was assembled with the shearing pins screwed in. The sample was transferred to the shear box in three layers (with hammering, if necessary).With the top plates fixed on the shear box, it was then transferred to the loading frame. The weights were then attached to the loading frame and the dial gauges set to zero. The machine was started and the proving ring readings were taken up to failure of the sample. The test was repeated for different weights (normal stress) and the observations were recorded.

Fig 3.13 Sample of OB + 15% Flyash

Fig 3.14 Addition of Water to OB + 15% Flyash Sample

The variation of shear stress with normal stress has been shown here. The detailed calculation of shear stress from the proving ring readings have been included in Appendix-I.

Sample: OB

Table 3.9: Normal Stress vs Shear Stress for OB Sample

Normal stress applied N, kg/cm^2	Shear stress τ, kg/cm^2
0.5	0.269
1.0	0.676
1.5	0.839
2.0	1.183
2.5	1.596

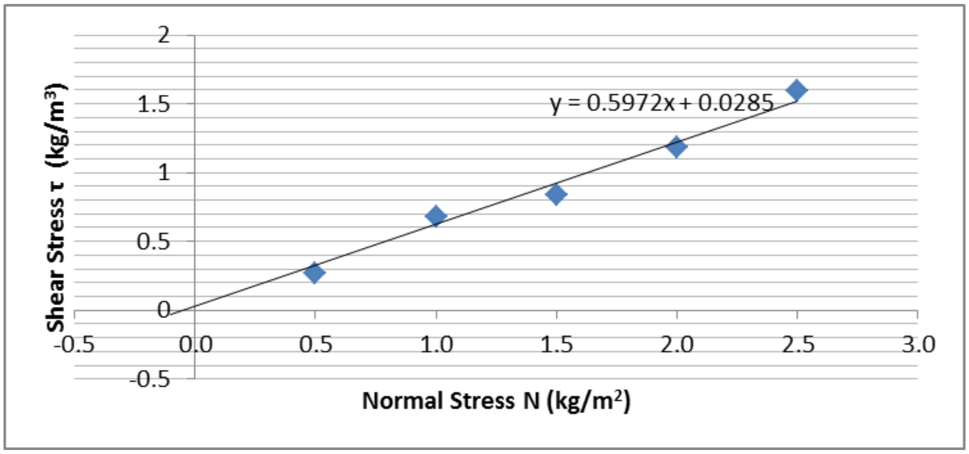

Fig. 3.15: Normal Stress vs Shear Stress for OB Sample

From the graph, cohesion = y intercept of the line = 0.0285 kg/cm^2 = 2705.85 Pa
Angle of Internal Friction = slope of the line = arctan (0.597) = 30.84°.

OB + 15% flyash

Table 3.10: Normal Stress vs Shear Stress for OB + 15% Flyash Sample

Normal stress applied N, kg/cm^2	Shear stress τ , kg/cm^2
0.50	0.378
1.00	0.525
1.50	0.660
2.00	1.117
2.50	1.262

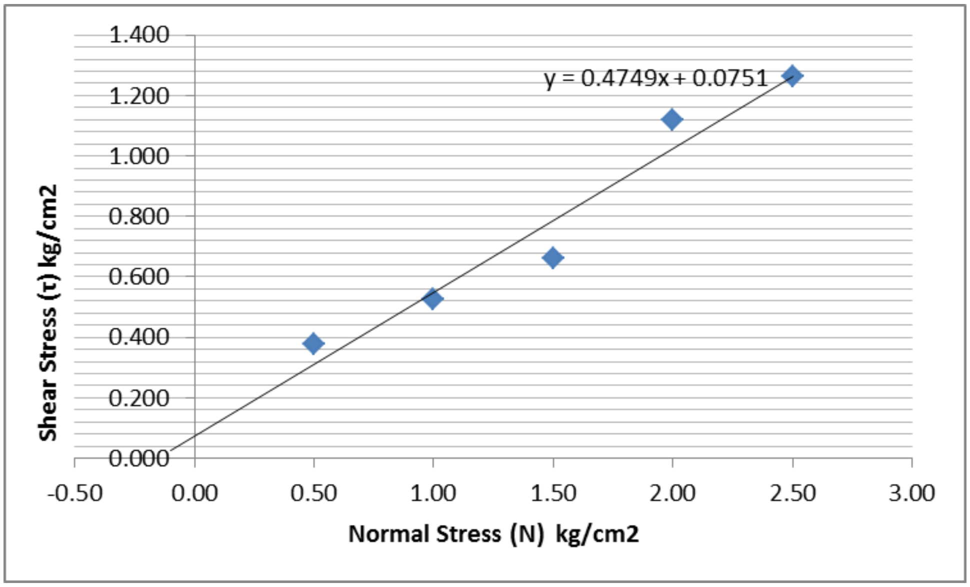

Fig. 3.16: Normal Stress vs Shear Stress for OB + 15% Flyash Sample

From the graph, cohesion = y intercept of the line = 0.0751 kg/cm^2 = 7367.31 Pa
Angle of Internal Friction = slope of the line = arctan (0.479) = 25.59°

OB +30% flyash

Table 3.11: Normal Stress vs Shear Stress for OB + 30% Flyash Sample

Normal stress applied N, kg/cm^2	Shear stress τ , kg/cm^2
0.5	0.364
1.0	0.538
1.5	0.867
2.0	1.159
2.5	1.246

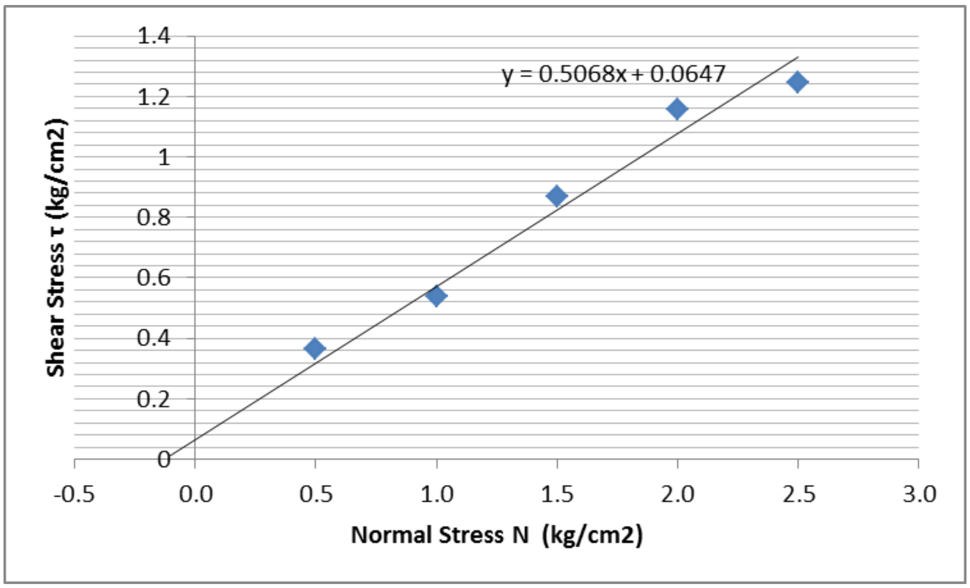

Fig. 3.17: Normal Stress vs Shear Stress for OB + 30% Flyash Sample

From the graph, cohesion = y intercept of the line = 0.0647 kg/cm^2 = 6347.07 Pa

Angle of Internal Friction = slope of the line = arctan (0.506) = 26.87°

Fig. 3.18: Failure Profile of the Samples

Fig. 3.19: Direct Shear Test Apparatus

The values of cohesion and angle of internal friction are found to be as follows:

Table 3.12: Cohesion and Angle of Internal Friction Values for Different Samples

Sample	Cohesion (kg/cm^2)	Friction Angle (°)
OB	0.0285	30.84
OB + 15% fly ash	0.0751	25.59
OB + 30% fly ash	0.0647	26.87

3.4 Overview of Numerical Modeling FLAC SLOPE

FLAC/Slope is a mini-version of FLAC that is designed specifically to perform factor-of-safety calculations for slope stability analysis. This version is operated entirely from FLAC's graphical interface (the GIIC) which provides for rapid creation of models for soil and/or rock slopes and solution of their stability condition.

FLAC/Slope provides an alternative to traditional "limit equilibrium" programs to determine factor of safety. Limit equilibrium codes use an approximate scheme, typically based on the method of slices, in which a number of assumptions are made (e.g., the location and angle of interslice forces). Several assumed failure surfaces are tested, and the one giving the lowest factor of safety is chosen. Equilibrium is only satisfied on an idealized set of surfaces. [16]

In contrast, FLAC/Slope provides a full solution of the coupled stress/displacement, equilibrium and constitutive equations. Given a set of properties, the system is determined to be stable or unstable By automatically performing a series of simulations while changing the strength properties, the factor of safety can be found to correspond to the point of stability, and the critical failure (slip) surface can be located.

FLAC/Slope does take longer to determine a factor of safety than a limit equilibrium program. However, with the advancement of computer processing speeds (e.g., 1 GHz and faster chips), solutions can now be obtained in a reasonable amount of time. This makes FLAC/Slope a practical alternative to a limit equilibrium program, and provides the following advantages over a limit equilibrium solution

- Any failure mode develops naturally; there is no need to specify a range of trial surfaces in advance.
- No artificial parameters (e.g., functions for interslice force angles) need to be given as input.
- Multiple failure surfaces (or complex internal yielding) evolve naturally, if the conditions give rise to them.
- Structural interaction (e.g., rock bolt, soil nail or geogrid) is modeled realistically as fully coupled deforming elements, not simply as equivalent forces.
- The solution consists of mechanisms that are kinematically feasible. (Note that the limit equilibrium method only considers forces, not kinematics.)

3.5 Procedure of Analysis

FLAC/Slope is specifically designed to perform multiple analyses and parametric studies for slope stability projects. The structure of the program allows different models in a project to be easily created, stored and accessed for direct comparison of model results. A FLAC/Slope analysis project is divided into four stages which is described below:

- **Stage 1: Defining a Model**

Each model in a project is named and listed in a tabbed bar in the Models stage. This allows easy access to any model and results in a project. New models can be added to the tabbed bar or deleted from it at any time in the project study. Models can also be restored (loaded) from previous projects and added to the current project. The slope boundary is also defined for each model at this stage.

- **Stage 2: Building the Model**

For a specific model, the slope conditions are defined in the Build stage. This includes: changes to the slope geometry, addition of layers, specification of materials and weak plane, application of surface loading, positioning of a water table and installation of reinforcement. Also, spatial regions of the model can be excluded from the factor-of-safety calculation. The build-stage conditions can be added, deleted and modified at any time during this stage.

- **Stage 3: Solving the Model**

In the Solve stage, the factor of safety is calculated. The resolution of the numerical mesh is selected first (coarse, medium and fine), and then the factor-of-safety calculation is performed. Different strength parameters can be selected for inclusion in the strength reduction approach to calculate the safety factor. By default, the material cohesion and friction angle are used.

- **Stage 4: Plotting the Result**

After the solution is complete, several output selections are available in the Plot stage for displaying the failure surface and recording the results. Model results are available for subsequent access and comparison to other models in the project. All models created within a project, along with their solutions, can be saved, the project files can be easily restored and results viewed at a later time.

3.6 Modeling the KTK OC mine dumps

3.6.1 Design Specifications

Soil dumps of height 30 m were simulated for the different mixtures of OB and flyash to find out the safe slope angle. The dumps were assumed to be dry and resting on a sandstone block whose dimensions were large enough in comparison to the dump so as not to affect its stability. The geo-technical parameters of the soils determined in the experimental analysis were used to generate the model of the dump.

Evaluation of the Factor of Safety (FoS) of the models were started at trial angles of 25° and 30°. The FoS = 1.2 was found to be lying within the FoS figures for 25° an 30° angles for all three dumps. The models were then evaluated for the interlaying angles i.e 26°, 27°, 28°, 29°. The steepest angle for which the FoS > 1.2 was accepted as the safe slope angle.

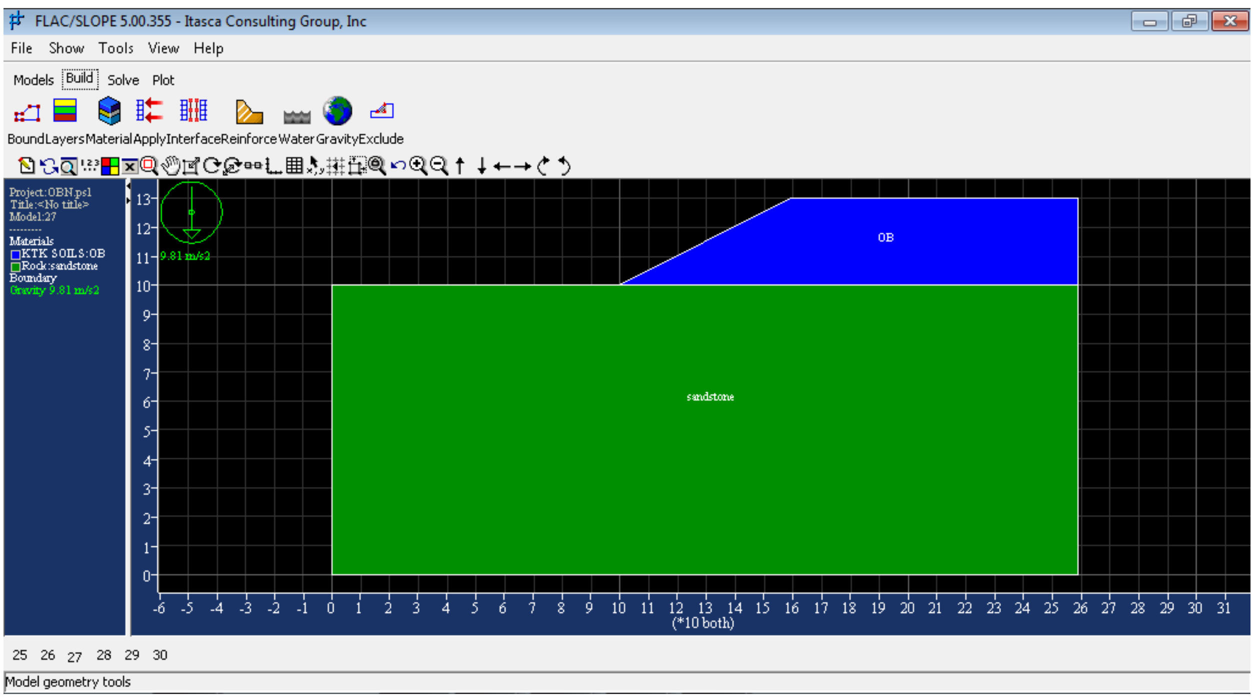

Fig. 3.20: FLAC SLOPE Interface Showing a Model

3.6.2 Sample: OB

The factor of safety obtained for different angles of the OB sample are shown in Table 4.1

Table 3.13: FoS for Different Angles of OB Dump

Angle (°)	FoS
25	1.42
26	1.36
27	1.31
28	1.26
29	**1.22**
30	1.17

From the above results, slope angle of 29° had the required factor of safety for OB dump.

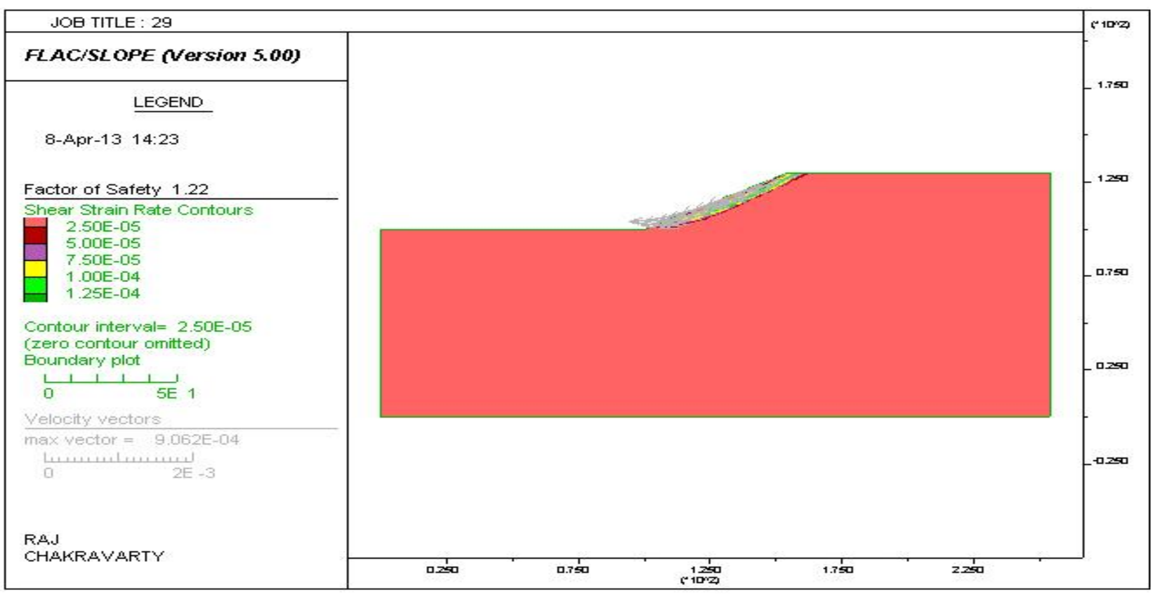

Fig. 3.21: FoS Plot for OB Dump with 29° Slope Angle

3.6.3 Sample: OB + 15% flyash

The factor of safety obtained for different angles of the OB+15% flyash are shown in Table 4.2

Table 3.14: FoS for Different Angles of OB + 15 % Flyash

Angle (°)	FoS
25	1.27
26	**1.22**
27	1.17
28	1.13
29	1.09
30	1.06

From the above results, slope angle of 26° had the required factor of safety for OB+15% flyash

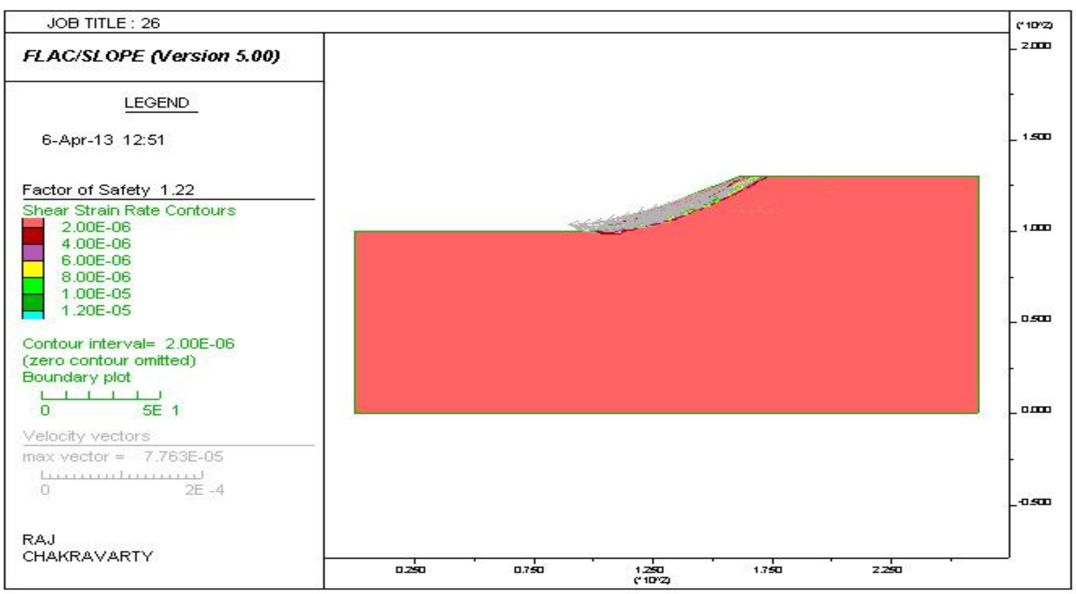

Fig. 3.22: FoS Plot for OB+15% Flyash with 26° Slope Angle

3.6.4 Sample: OB + 30% flyash

The factor of safety obtained for different angles of the OB+30% flyash are shown in Table 4.3.

Table 3.15:_FoS for Different Angles of OB + 30% Flyash

Angle (°)	FoS
25	1.40
26	1.33
27	1.27
28	**1.22**
29	1.17
30	1.12

From the above results, slope angle of 28° had the required factor of safety for OB+30% flyash

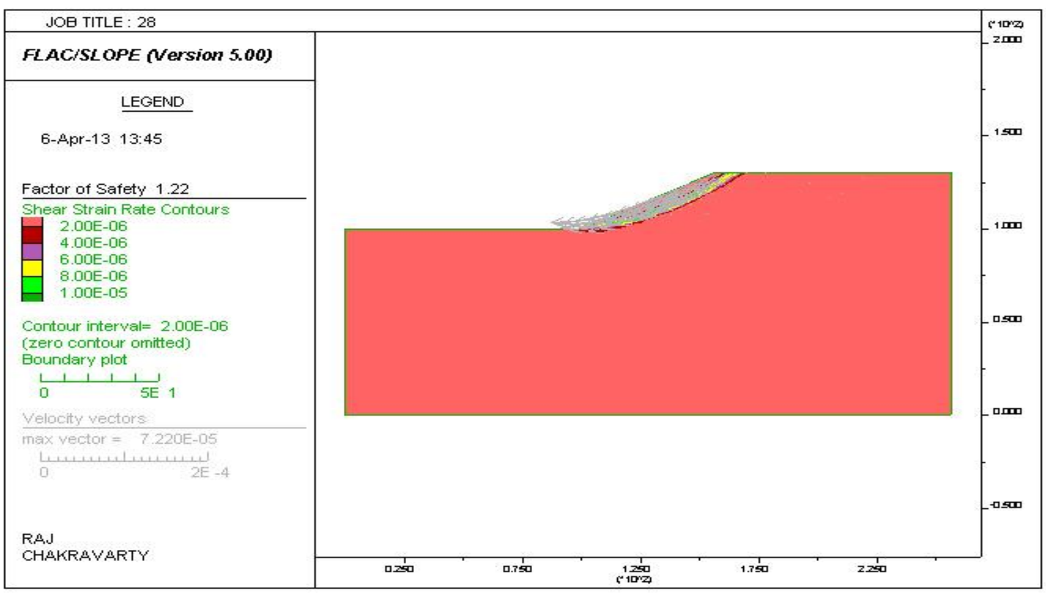

Fig. 3.23: FoS Plot for OB + 30% Flyash Dump with 28° Slope Angle

CHAPTER 4

CONCLUSION

CONCLUSION

The different geo-technical parameters of the OB and flyash mixtures i.e. density, cohesion and friction angle values were used to model the dumps in FLAC SLOPE software.

The slope angles which were found to be safe i.e. FOS > 1.2, for the different mixtures of OB and fly ash are shown in table below:

Soil	**Angle (°)**
OB	29
OB + 15% fly ash	26
OB + 30% fly ash	28

- The initial decrease in slope angle from 29º to 26º with the addition of 15% flyash might be attributed to the inadequate packing of voids between OB particles by the finer sized flyash particles
- With increasing quantity of flyash i.e. at 30%, packing of the voids would become more compact as they reduce the void ratio. This would lead to the increase in slope angle obtained with OB + 30% fly ash from 26 º to 28º. However, there was no significant change in slope angle with addition of flyash vis-à-vis OB.

4.1 Scope for Future Work

- Other percentages of fly ash can be mixed with OB to obtain a detailed study of the variation of geo-technical parameters as well the factor of safety of the resulting dumps.

- Three dimensional models of the dumps should also be evaluated as they allow the modeling of more complex geo-mining conditions than the two dimensional models.

- The method of mixing of fly ash with OB such as adding them in alternate layers, can be explored further.
- As the dumps were modeled in a dry condition the effect of groundwater and rainfall on the slopes can be examined.

REFERENCES

1. Lokeshappa, B and Dikshit, Anil Kumar, (2011), "*Disposal and Management of Flyash*", International Conference on Life Science and Technology (IPCBEE), Vol.3, IACSIT Press, Singapore, pp: 1.
2. Ministry of Environment and Forests, (2008), "*Draft Fly Ash Amendment Notification*", New Delhi, pp: 9.
3. Singh, R.D, (2010), "*Principles and Practices of Modern Coal Mining*", New Age International, New Delhi, pp: 616-617.
4. Bauer, A. & Calder, P.N. (1971), "*The Influence and Evaluation of Blasting on Stability*", In Stability in Open Pit Mining, Proc. 1st International Conference on Stability in Open Pit Mining (Vancouver, November 23-25, 1970), and New York: Society of Mining Engineers, A.I.M.E, pp. 83-94.
5. Blasio, F.V, (2011), "*Introduction to the Physics of Landslides: Lecture Notes on the Dynamics of Mass Wasting*", Springer Science and Business Media, London, pp: 30-31.
6. Willie, Duncan C. and Mah, Christopher W. (2005), "*Rock Slope Engineering: Civil and Mining*", 4th Edition, Taylor and Francis, London, pp: 11 – 15.
7. Jhanwar, J.C. (2008), "*Slope Stability Scenario of Opencast Mines in India*", Lecture Note, Central Institute of Mining an Fuel Research Regional Centre, Nagpur, pp: 1-2.
8. Piteau, D.R. and Peckover, F.L., (1978), *"Engineering of Rock Slopes. In Special Report 176:Landslides: Analysis and Control.",* TRB, National Research Council, Washington, D.C., pp.192-234.
9. Hoek, E. and Bray, J.W., (1981)*, "Rock Slope Engineering",* 3rd Edition, Institution of Mining and Metallurgy, London, pp: 402.
10. Goodman, R.E. and Bray, J.W., 1976, "*Toppling of Rock Slopes",* Specialty Conference on Rock Engineering for Foundations and Slopes, Boulder, Colo., American Society of Civil Engineers, New York, Vol.2, pp: 201-234.
11. The Coal Mine Regulations,1957, Chapter X, Section 98.
12. IS: 2720, (1986), "*Method of Test for Soils, Part 13:Direct Shear Test* ", New Delhi, pp: 3-10.

13. IS: 2720, (1980), "*Method of Test for Soils, Part 7: Determination of water content-dry density relation using light compaction* ", New Delhi, pp: 3-7.
14. IS: 2720, (1985), "*Method of Test for Soils, Part 4:Grain Size Analysis*", New Delhi, pp: 3-8.
15. Das, G. (2011), "*Analysis of Slope Stability for Waste Dumps in a mine*", B.Tech Thesis, pp: 22-30
16. FLAC SLOPE Users Guide, (2002), Itasca Consulting Group, Minnesota, pp: 2-5.
17. Jayanthu, S., Das, S.K. and Equeenuddin, S.K., (2012), "*Stability of Fly Ash and Overbuden Material as Back Filling in Opencast Mines –A Case Study*", International Conference on Chemical, Civil and Environment engineering, Dubai, pp: 276-278.
18. Chaulya, S.K., Singh, R.S., Chakraborty, M.K., Dhar, B.B., (1999), "*Numerical Modelling of Biostabilisation for a Coal Mine Overburden Dump Slope"*, Ecological Modelling, Elsevier Science Press, London, pp: 5-12.
19. Singh, V.K. and Singh, J.K. and Kumar, Ajit., (2004). "*Geotechnical Studies and optimal slope design at Lajkura Opencast coal mine*", International Journal of Rock Mechanics and Mining Sciences pp. 524.
20. Kainthola, A., Verma, D., Gupte, S., Singh, T.N., (2011), "*A Coal Mine Dump Stability Analysis: Case Study*", Geomaterials, London, pp: 3-13.

APPENDIX – I:

Shear Stress Calculation from Proving Ring readings for Different Normal Stress

Table A1: Shear Stress calculation for OB with 0.5 kg/cm^2 Load

Sample = OB

Density (ρ) = 2.02 g/cc
Moisture (M) = 9.16 %

Mould dimensions = 6 x 6 x 2.8 cc
Area of the mould (A0) = 36 cm2
Volume of the mould (V) = 100.8 cc

Amount of sample taken = V x d = 203.61 gm
Water added = 9.16% of 203.61 = 18.65 ml

Vertical load = 0.5 kg/cm2
Rate of shearing = 1.25 mm/min

Calibration of proving ring: 199 ↔ 0.50 kN (50.96 kg)

Strain Gauge dial reading (S)	Proving ring reading (P)	LC of Strain gauge (mm)	displacement d = (S*LC)/10 (cm)	Corrected Area A1 = A0(1-d/3) cm2	Applied load L = (P*50.96)/199 kg	Shearing Stress τ = L / A1 kg/cm2
20	5	0.01	0.02	35.76	1.280	0.036
40	9	0.01	0.04	35.52	2.305	0.065
60	11	0.01	0.06	35.28	2.817	0.080
80	14	0.01	0.08	35.04	3.585	0.102
100	17	0.01	0.10	34.80	4.353	0.125
120	19	0.01	0.12	34.56	4.866	0.141
140	22	0.01	0.14	34.32	5.634	0.164
160	25	0.01	0.16	34.08	6.402	0.188
180	26	0.01	0.18	33.84	6.658	0.197
200	26	0.01	0.20	33.60	6.658	0.198
220	27	0.01	0.22	33.36	6.914	0.207
240	28	0.01	0.24	33.12	7.170	0.216
260	29	0.01	0.26	32.88	7.426	0.226
280	30	0.01	0.28	32.64	7.682	0.235
300	31	0.01	0.30	32.40	7.938	0.245
320	31	0.01	0.32	32.16	7.938	0.247
340	32	0.01	0.34	31.92	8.195	0.257
360	32	0.01	0.36	31.68	8.195	0.259
380	**33**	0.01	0.38	31.44	8.451	0.269
400	32	0.01	0.40	31.20	8.195	0.263
420	32	0.01	0.42	30.96	8.195	0.265
440	31	0.01	0.44	30.72	7.938	0.258
460	30	0.01	0.46	30.48	7.682	0.252
480	30	0.01	0.48	30.24	7.682	0.254
500	29	0.01	0.50	30.00	7.426	0.248
						0.269

Table A2: Shear Stress calculation for OB with 1.0 kg/cm^2 Load

Sample = OB

Density (ρ) = 2.02 g/cc
Moisture (M) = 9.16 %

Mould dimensions = 6 x 6 x 2.8 cc
Area of the mould (A0) = 36 cm2
Volume of the mould (V) = 100.8 cc

Amount of sample taken = V x d = 203.61 gm
Water added = 9.16% of 203.61 = 18.65 ml

Vertical load = 1.0 kg/cm2
Rate of shearing = 1.25 mm/min

Calibration of proving ring: 199 ↔ 0.50 kN (50.96 kg)

Strain Gauge dial reading (S)	Proving ring reading (P)	LC of Strain gauge (mm)	displacement d = (S*LC)/10 (cm)	Corrected Area A1 = A0(1-d/3) cm2	Applied load L = (P*50.96)/199 kg	Shearing Stress τ = L / A1 kg/cm2
20	28	0.01	0.02	35.76	7.170	0.201
40	40	0.01	0.04	35.52	10.243	0.288
60	49	0.01	0.06	35.28	12.548	0.356
80	56	0.01	0.08	35.04	14.341	0.409
100	61	0.01	0.10	34.80	15.621	0.449
120	65	0.01	0.12	34.56	16.645	0.482
140	69	0.01	0.14	34.32	17.670	0.515
160	72	0.01	0.16	34.08	18.438	0.541
180	75	0.01	0.18	33.84	19.206	0.568
200	78	0.01	0.20	33.60	19.974	0.594
220	80	0.01	0.22	33.36	20.486	0.614
240	81	0.01	0.24	33.12	20.743	0.626
260	82	0.01	0.26	32.88	20.999	0.639
280	82	0.01	0.28	32.64	20.999	0.643
300	83	0.01	0.30	32.40	21.255	0.656
320	84	0.01	0.32	32.16	21.511	0.669
340	84	0.01	0.34	31.92	21.511	0.674
360	83	0.01	0.36	31.68	21.255	0.671
380	83	0.01	0.38	31.44	21.255	0.676
400	82	0.01	0.40	31.20	20.999	0.673
420	81	0.01	0.42	30.96	20.743	0.670
440	80	0.01	0.44	30.72	20.486	0.667
460	78	0.01	0.46	30.48	19.974	0.655
480	77	0.01	0.48	30.24	19.718	0.652
500	75	0.01	0.50	30.00	19.206	0.640
						0.676

Table A3: Shear Stress calculation for OB with 1.5 kg/cm^2 Load

Sample = OB

Density (ρ) = 2.02 g/cc
Moisture (M) = 9.16 %

Mould dimensions = 6 x 6 x 2.8 cc
Area of the mould (A0) = 36 cm2
Volume of the mould (V) = 100.8 cc

Amount of sample taken = V x d = 203.61 gm
Water added = 9.16% of 203.61 = 18.65 ml

Vertical load = 1.5 kg/cm2
Rate of shearing = 1.25 mm/min

Calibration of proving ring: 199 ↔ 0.50 kN (50.96 kg)

Strain Gauge dial reading (S)	Proving ring reading (P)	LC of Strain gauge (mm)	displacement d = (S*LC)/10 (cm)	Corrected Area A1 = A0(1-d/3) cm2	Applied load L = (P*50.96)/199 kg	Shearing Stress τ = L / A1 kg/cm2
20	21	0.01	0.02	35.76	5.378	0.150
40	32	0.01	0.04	35.52	8.195	0.231
60	43	0.01	0.06	35.28	11.011	0.312
80	52	0.01	0.08	35.04	13.316	0.380
100	63	0.01	0.10	34.80	16.133	0.464
120	72	0.01	0.12	34.56	18.438	0.534
140	80	0.01	0.14	34.32	20.486	0.597
160	86	0.01	0.16	34.08	22.023	0.646
180	90	0.01	0.18	33.84	23.047	0.681
200	93	0.01	0.20	33.60	23.815	0.709
220	97	0.01	0.22	33.36	24.840	0.745
240	100	0.01	0.24	33.12	25.608	0.773
260	101	0.01	0.26	32.88	25.864	0.787
280	102	0.01	0.28	32.64	26.120	0.800
300	103	0.01	0.30	32.40	26.376	0.814
320	104	0.01	0.32	32.16	26.632	0.828
340	104	0.01	0.34	31.92	26.632	0.834
360	103	0.01	0.36	31.68	26.376	0.833
380	103	0.01	0.38	31.44	26.376	0.839
400	102	0.01	0.40	31.20	26.120	0.837
420	101	0.01	0.42	30.96	25.864	0.835
440	100	0.01	0.44	30.72	25.608	0.834
460	98	0.01	0.46	30.48	25.096	0.823
480	97	0.01	0.48	30.24	24.840	0.821
500	97	0.01	0.50	30.00	24.840	0.828
						0.839

Table A4: Shear Stress Calculation for OB with 2.0 kg/cm^2 Load

Sample = OB

Density (ρ) = 2.02 g/cc
Moisture (M) = 9.16 %

Mould dimensions = 6 x 6 x 2.8 cc
Area of the mould (A0) = 36 cm2
Volume of the mould (V) = 100.8 cc

Amount of sample taken = V x d = 203.61 gm
Water added = 9.16% of 203.61 = 18.65 ml

Vertical load = 2.0 kg/cm2
Rate of shearing = 1.25 mm/min

Calibration of proving ring: 199 ↔ 0.50 kN (50.96 kg)

Strain Gauge dial reading (S)	Proving ring reading (P)	LC of Strain gauge (mm)	displacement d = (S*LC)/10 (cm)	Corrected Area A1 = A0(1-d/3) cm2	Applied load L = (P*50.96)/199 kg	Shearing Stress τ = L / A1 kg/cm2
20	25	0.01	0.02	35.76	6.402	0.179
40	40	0.01	0.04	35.52	10.243	0.288
60	57	0.01	0.06	35.28	14.597	0.414
80	73	0.01	0.08	35.04	18.694	0.534
100	88	0.01	0.10	34.80	22.535	0.648
120	96	0.01	0.12	34.56	24.584	0.711
140	101	0.01	0.14	34.32	25.864	0.754
160	106	0.01	0.16	34.08	27.145	0.796
180	116	0.01	0.18	33.84	29.705	0.878
200	125	0.01	0.20	33.60	32.010	0.953
220	131	0.01	0.22	33.36	33.547	1.006
240	136	0.01	0.24	33.12	34.827	1.052
260	141	0.01	0.26	32.88	36.107	1.098
280	143	0.01	0.28	32.64	36.619	1.122
300	144	0.01	0.30	32.40	36.876	1.138
320	145	0.01	0.32	32.16	37.132	1.155
340	146	0.01	0.34	31.92	37.388	1.171
360	146	0.01	0.36	31.68	37.388	1.180
380	145	0.01	0.38	31.44	37.132	1.181
400	144	0.01	0.40	31.20	36.876	1.182
420	143	0.01	0.42	30.96	36.619	1.183
440	141	0.01	0.44	30.72	36.107	1.175
460	138	0.01	0.46	30.48	35.339	1.159
480	136	0.01	0.48	30.24	34.827	1.152
500	133	0.01	0.50	30.00	34.059	1.135
						1.183

Table A5: Shear Stress Calculation for OB with 2.5 kg/cm^2 Load

Sample = OB

Density (ρ) = 2.02 g/cc
Moisture (M) = 9.16 %

Mould dimensions = 6 x 6 x 2.8 cc
Area of the mould (A0) = 36 cm2
Volume of the mould (V) = 100.8 cc

Amount of sample taken = V x d = 203.61 gm
Water added = 9.16% of 203.61 = 18.65 ml

Vertical load = 2.5 kg/cm2
Rate of shearing = 1.25 mm/min

Calibration of proving ring: 199 ↔ 0.50 kN (50.96 kg)

Strain Gauge dial reading (S)	Proving ring reading (P)	LC of Strain gauge (mm)	displacement d = (S*LC)/10 (cm)	Corrected Area A1 = A0(1-d/3) cm2	Applied load L = (P*50.96)/199 kg	Shearing Stress τ = L / A1 kg/cm2
20	27	0.01	0.02	35.76	6.914	0.193
40	58	0.01	0.04	35.52	14.853	0.418
60	79	0.01	0.06	35.28	20.230	0.573
80	98	0.01	0.08	35.04	25.096	0.716
100	119	0.01	0.10	34.80	30.474	0.876
120	138	0.01	0.12	34.56	35.339	1.023
140	147	0.01	0.14	34.32	37.644	1.097
160	159	0.01	0.16	34.08	40.717	1.195
180	170	0.01	0.18	33.84	43.534	1.286
200	179	0.01	0.20	33.60	45.838	1.364
220	185	0.01	0.22	33.36	47.375	1.420
240	190	0.01	0.24	33.12	48.655	1.469
260	193	0.01	0.26	32.88	49.424	1.503
280	196	0.01	0.28	32.64	50.192	1.538
300	196	0.01	0.30	32.40	50.192	1.549
320	197	0.01	0.32	32.16	50.448	1.569
340	196	0.01	0.34	31.92	50.192	1.572
360	195	0.01	0.36	31.68	49.936	1.596
380	193	0.01	0.38	31.44	49.424	1.595
400	193	0.01	0.40	31.20	49.424	1.594
420	191	0.01	0.42	30.96	48.911	1.594
440	190	0.01	0.44	30.72	48.655	1.593
460	187	0.01	0.46	30.48	47.887	1.591
480	185	0.01	0.48	30.24	47.375	1.590
500	183	0.01	0.50	30.00	46.863	1.562
						1.596

Table A6: Shear Stress Calculation for OB+15% Flyash with 0.5 kg/cm^2 Load

Sample = OB + 15% flyash						
Density (ρ) = 1.91 g/cc						
Moisture (M) = 10.11 %						
Mould dimensions = 6 x 6 x 2.8 cc				Amount of sample taken = V x d = 192.52 gm		
Area of the mould (A0) = 36 cm2				Water added = 10.11% of 192.52 = 19.46 ml		
Volume of the mould (V) = 100.8 cc						
Vertical load = 0.5 kg/cm2				Calibration of proving ring: 199 ↔ 0.50 kN (50.96 kg)		
Rate of shearing = 1.25 mm/min						
Strain Gauge dial reading (S)	Proving ring reading (P)	LC of Strain gauge (mm)	displacement d = (S*LC)/10 (cm)	Corrected Area A1 = A0(1-d/3) cm2	Applied load L = (P*50.96)/199 kg	Shearing Stress τ = L / A1 kg/cm2
20	5	0.01	0.02	35.76	1.280	0.036
40	7	0.01	0.04	35.52	1.793	0.050
60	9	0.01	0.06	35.28	2.305	0.065
80	10	0.01	0.08	35.04	2.561	0.073
100	13	0.01	0.10	34.80	3.329	0.096
120	15	0.01	0.12	34.56	3.841	0.111
140	17	0.01	0.14	34.32	4.353	0.127
160	20	0.01	0.16	34.08	5.122	0.150
180	22	0.01	0.18	33.84	5.634	0.166
200	27	0.01	0.20	33.60	6.914	0.206
220	31	0.01	0.22	33.36	7.938	0.238
240	35	0.01	0.24	33.12	8.963	0.271
260	38	0.01	0.26	32.88	9.731	0.296
280	40	0.01	0.28	32.64	10.243	0.314
300	42	0.01	0.30	32.40	10.755	0.332
320	43	0.01	0.32	32.16	11.011	0.342
340	45	0.01	0.34	31.92	11.524	0.361
360	45	0.01	0.36	31.68	11.524	0.364
380	46	0.01	0.38	31.44	11.780	0.375
400	46	0.01	0.40	31.20	11.780	0.378
420	45	0.01	0.42	30.96	11.524	0.372
440	45	0.01	0.44	30.72	11.524	0.375
460	44	0.01	0.46	30.48	11.268	0.370
480	43	0.01	0.48	30.24	11.011	0.364
500	43	0.01	0.50	30.00	11.011	0.367
						0.378

Table A7: Shear Stress Calculation for OB+15% Flyash with 1.0 kg/cm^2 Load

Sample = OB + 15% flyash

Density (ρ) = 1.91 g/cc
Moisture (M) = 10.11 %

Mould dimensions = 6 x 6 x 2.8 cc
Area of the mould (A0) = 36 cm2
Volume of the mould (V) = 100.8 cc

Amount of sample taken = V x d = 192.52 gm
Water added = 10.11% of 192.52 = 19.46 ml

Vertical load = 1.0 kg/cm2
Rate of shearing = 1.25 mm/min

Calibration of proving ring: 199 ↔ 0.50 kN (50.96 kg)

Strain Gauge dial reading (S)	Proving ring reading (P)	LC of Strain gauge (mm)	displacement d = (S*LC)/10 (cm)	Corrected Area A1 = A0(1-d/3) cm2	Applied load L = (P*50.96)/199 kg	Shearing Stress τ = L / A1 kg/cm2
20	11	0.01	0.02	35.76	2.817	0.079
40	17	0.01	0.04	35.52	4.353	0.123
60	22	0.01	0.06	35.28	5.634	0.160
80	26	0.01	0.08	35.04	6.658	0.190
100	31	0.01	0.10	34.80	7.938	0.228
120	36	0.01	0.12	34.56	9.219	0.267
140	42	0.01	0.14	34.32	10.755	0.313
160	47	0.01	0.16	34.08	12.036	0.353
180	53	0.01	0.18	33.84	13.572	0.401
200	58	0.01	0.20	33.60	14.853	0.442
220	60	0.01	0.22	33.36	15.365	0.461
240	61	0.01	0.24	33.12	15.621	0.472
260	62	0.01	0.26	32.88	15.877	0.483
280	62	0.01	0.28	32.64	15.877	0.486
300	63	0.01	0.30	32.40	16.133	0.498
320	64	0.01	0.32	32.16	16.389	0.510
340	64	0.01	0.34	31.92	16.389	0.513
360	65	0.01	0.36	31.68	16.645	0.525
380	64	0.01	0.38	31.44	16.389	0.521
400	64	0.01	0.40	31.20	16.389	0.525
420	63	0.01	0.42	30.96	16.133	0.521
440	62	0.01	0.44	30.72	15.877	0.517
460	61	0.01	0.46	30.48	15.621	0.512
480	60	0.01	0.48	30.24	15.365	0.508
500	60	0.01	0.50	30.00	15.365	0.512
						0.525

Table A8: Shear Stress Calculation for OB+15% Flyash with 1.5 kg/cm^2 Load

Sample = OB + 15% flyash

Density (ρ) = 1.91 g/cc
Moisture (M) = 10.11 %

Mould dimensions = 6 x 6 x 2.8 cc
Area of the mould (A0) = 36 cm2
Volume of the mould (V) = 100.8 cc

Amount of sample taken = V x d = 192.52 gm
Water added = 10.11% of 192.52 = 19.46 ml

Vertical load = 1.5 kg/cm2
Rate of shearing = 1.25 mm/min

Calibration of proving ring: 199 ↔ 0.50 kN (50.96 kg)

Strain Gauge dial reading (S)	Proving ring reading (P)	LC of Strain gauge (mm)	displacement d = (S*LC)/10 (cm)	Corrected Area A1 = A0(1-d/3) cm2	Applied load L = (P*50.96)/199 kg	Shearing Stress τ = L / A1 kg/cm2
20	15	0.01	0.02	35.76	3.841	0.107
40	18	0.01	0.04	35.52	4.609	0.130
60	20	0.01	0.06	35.28	5.122	0.145
80	22	0.01	0.08	35.04	5.634	0.161
100	29	0.01	0.10	34.80	7.426	0.213
120	35	0.01	0.12	34.56	8.963	0.259
140	42	0.01	0.14	34.32	10.755	0.313
160	49	0.01	0.16	34.08	12.548	0.368
180	53	0.01	0.18	33.84	13.572	0.401
200	60	0.01	0.20	33.60	15.365	0.457
220	65	0.01	0.22	33.36	16.645	0.499
240	71	0.01	0.24	33.12	18.182	0.549
260	75	0.01	0.26	32.88	19.206	0.584
280	78	0.01	0.28	32.64	19.974	0.612
300	79	0.01	0.30	32.40	20.230	0.624
320	80	0.01	0.32	32.16	20.486	0.637
340	80	0.01	0.34	31.92	20.486	0.642
360	81	0.01	0.36	31.68	20.743	0.655
380	81	0.01	0.38	31.44	20.743	0.660
400	80	0.01	0.40	31.20	20.486	0.657
420	79	0.01	0.42	30.96	20.230	0.653
440	79	0.01	0.44	30.72	20.230	0.659
460	78	0.01	0.46	30.48	19.974	0.655
480	77	0.01	0.48	30.24	19.718	0.652
500	77	0.01	0.50	30.00	19.718	0.657
						0.660

Table A9: Shear Stress Calculation for OB+15% Flyash with 2.0 kg/cm^2 Load

Sample = OB + 15% flyash

Density (ρ) = 1.91 g/cc
Moisture (M) = 10.11 %

Mould dimensions = 6 x 6 x 2.8 cc
Area of the mould (A0) = 36 cm2
Volume of the mould (V) = 100.8 cc

Amount of sample taken = V x d = 192.52 gm
Water added = 10.11% of 192.52 = 19.46 ml

Vertical load = 2.0 kg/cm2
Rate of shearing = 1.25 mm/min

Calibration of proving ring: 199 ↔ 0.50 kN (50.96 kg)

Strain Gauge dial reading (S)	Proving ring reading (P)	LC of Strain gauge (mm)	displacement d = (S*LC)/10 (cm)	Corrected Area A1 = A0(1-d/3) cm2	Applied load L = (P*50.96)/199 kg	Shearing Stress τ = L / A1 kg/cm2
20	27	0.01	0.02	35.76	6.914	0.193
40	46	0.01	0.04	35.52	11.780	0.332
60	59	0.01	0.06	35.28	15.109	0.428
80	70	0.01	0.08	35.04	17.926	0.512
100	79	0.01	0.10	34.80	20.230	0.581
120	87	0.01	0.12	34.56	22.279	0.645
140	95	0.01	0.14	34.32	24.328	0.709
160	103	0.01	0.16	34.08	26.376	0.774
180	109	0.01	0.18	33.84	27.913	0.825
200	114	0.01	0.20	33.60	29.193	0.869
220	120	0.01	0.22	33.36	30.730	0.921
240	125	0.01	0.24	33.12	32.010	0.966
260	130	0.01	0.26	32.88	33.290	1.012
280	134	0.01	0.28	32.64	34.315	1.051
300	136	0.01	0.30	32.40	34.827	1.075
320	137	0.01	0.32	32.16	35.083	1.091
340	138	0.01	0.34	31.92	35.339	1.107
360	138	0.01	0.36	31.68	35.339	1.116
380	137	0.01	0.38	31.44	35.083	1.116
400	135	0.01	0.40	31.20	34.571	1.108
420	135	0.01	0.42	30.96	34.571	1.117
440	133	0.01	0.44	30.72	34.059	1.109
460	132	0.01	0.46	30.48	33.803	1.109
480	131	0.01	0.48	30.24	33.547	1.109
500	129	0.01	0.50	30.00	33.034	1.101
						1.117

Table A10: Shear Stress Calculation for OB+15% Flyash with 2.5 kg/cm^2 Load

Sample = OB + 15% flyash

Density (ρ) = 1.91 g/cc
Moisture (M) = 10.11 %

Mould dimensions = 6 x 6 x 2.8 cc
Area of the mould (A0) = 36 cm2
Volume of the mould (V) = 100.8 cc

Amount of sample taken = V x d = 192.52 gm
Water added = 10.11% of 192.52 = 19.46 ml

Vertical load = 2.5 kg/cm2
Rate of shearing = 1.25 mm/min

Calibration of proving ring: 199 ↔ 0.50 kN (50.96 kg)

Strain Gauge dial reading (S)	Proving ring reading (P)	LC of Strain gauge (mm)	displacement d = (S*LC)/10 (cm)	Corrected Area A1 = A0(1-d/3) cm2	Applied load L = (P*50.96)/199 kg	Shearing Stress τ = L / A1 kg/cm2
20	29	0.01	0.02	35.76	7.544	0.211
40	50	0.01	0.04	35.52	13.007	0.366
60	61	0.01	0.06	35.28	15.868	0.450
80	74	0.01	0.08	35.04	19.250	0.549
100	86	0.01	0.10	34.80	22.371	0.643
120	99	0.01	0.12	34.56	25.753	0.745
140	104	0.01	0.14	34.32	27.054	0.788
160	113	0.01	0.16	34.08	29.395	0.863
180	121	0.01	0.18	33.84	31.476	0.930
200	130	0.01	0.20	33.60	33.817	1.006
220	139	0.01	0.22	33.36	36.158	1.084
240	145	0.01	0.24	33.12	37.719	1.139
260	149	0.01	0.26	32.88	38.760	1.179
280	151	0.01	0.28	32.64	39.280	1.203
300	151	0.01	0.30	32.40	39.280	1.212
320	152	0.01	0.32	32.16	39.540	1.229
340	153	0.01	0.34	31.92	39.800	1.247
360	152	0.01	0.36	31.68	39.540	1.248
380	152	0.01	0.38	31.44	39.540	1.258
400	151	0.01	0.40	31.20	39.280	1.259
420	150	0.01	0.42	30.96	39.020	1.260
440	149	0.01	0.44	30.72	38.760	1.262
460	147	0.01	0.46	30.48	38.240	1.255
480	145	0.01	0.48	30.24	37.719	1.247
500	144	0.01	0.50	30.00	37.459	1.249
						1.262

Table A11: Shear Stress Calculation for OB+30% Flyash with 0.5 kg/cm^2 Load

Sample = OB + 30% flyash

Density (ρ) = 1.70 g/cc
Moisture (M) = 15.95 %

Mould dimensions = 6 x 6 x 2.8 cc
Area of the mould (A0) = 36 cm2
Volume of the mould (V) = 100.8 cc

Amount of sample taken = V x ρ = 171.36 gm
Water added = 15.95% of 171.36 = 27.33 ml

Vertical load = 0.5 kg/cm2
Rate of shearing = 1.25 mm/min

Calibration of proving ring: 199 ↔ 0.50 kN (50.96 kg)

Strain Gauge dial reading (S)	Proving ring reading (P)	LC of Strain gauge (mm)	displacement d = (S*LC)/10 (cm)	Corrected Area A1 = A0(1-d/3) cm2	Applied load L = (P*50.96)/199 kg	Shearing Stress τ = L / A1 kg/cm2
20	7	0.01	0.02	35.76	1.793	0.050
40	9	0.01	0.04	35.52	2.305	0.065
60	11	0.01	0.06	35.28	2.817	0.080
80	15	0.01	0.08	35.04	3.841	0.110
100	20	0.01	0.10	34.80	5.122	0.147
120	26	0.01	0.12	34.56	6.658	0.193
140	32	0.01	0.14	34.32	8.195	0.239
160	39	0.01	0.16	34.08	9.987	0.293
180	40	0.01	0.18	33.84	10.243	0.303
200	41	0.01	0.20	33.60	10.499	0.312
220	41	0.01	0.22	33.36	10.499	0.315
240	42	0.01	0.24	33.12	10.755	0.325
260	42	0.01	0.26	32.88	10.755	0.327
280	43	0.01	0.28	32.64	11.011	0.337
300	44	0.01	0.30	32.40	11.268	0.348
320	44	0.01	0.32	32.16	11.268	0.350
340	45	0.01	0.34	31.92	11.524	0.361
360	45	0.01	0.36	31.68	11.524	0.364
380	44	0.01	0.38	31.44	11.268	0.358
400	44	0.01	0.40	31.20	11.268	0.361
420	43	0.01	0.42	30.96	11.011	0.356
440	42	0.01	0.44	30.72	10.755	0.350
460	42	0.01	0.46	30.48	10.755	0.353
480	41	0.01	0.48	30.24	10.499	0.347
500	40	0.01	0.50	30.00	10.243	0.341
						0.364

Table A12:_Shear Stress Calculation for OB+30% Flyash with 1.0 kg/cm^2 Load

Sample = OB + 30% flyash

Density (ρ) = 1.70 g/cc
Moisture (M) = 15.95 %

Mould dimensions = 6 x 6 x 2.8 cc
Area of the mould (A0) = 36 cm2
Volume of the mould (V) = 100.8 cc

Amount of sample taken = V x ρ = 171.36 gm
Water added = 15.95% of 171.36 = 27.33 ml

Vertical load = 1.0 kg/cm2
Rate of shearing = 1.25 mm/min

Calibration of proving ring: 199 ↔ 0.50 kN (50.96 kg)

Strain Gauge dial reading (S)	Proving ring reading (P)	LC of Strain gauge (mm)	displacement d = (S*LC)/10 (cm)	Corrected Area A1 = A0(1-d/3) cm2	Applied load L = (P*50.96)/199 kg	Shearing Stress τ = L / A1 kg/cm2
20	9	0.01	0.02	35.76	2.305	0.064
40	15	0.01	0.04	35.52	3.841	0.108
60	20	0.01	0.06	35.28	5.122	0.145
80	26	0.01	0.08	35.04	6.658	0.190
100	30	0.01	0.10	34.80	7.682	0.221
120	34	0.01	0.12	34.56	8.707	0.252
140	40	0.01	0.14	34.32	10.243	0.298
160	45	0.01	0.16	34.08	11.524	0.338
180	51	0.01	0.18	33.84	13.060	0.386
200	56	0.01	0.20	33.60	14.341	0.427
220	60	0.01	0.22	33.36	15.365	0.461
240	61	0.01	0.24	33.12	15.621	0.472
260	62	0.01	0.26	32.88	15.877	0.483
280	62	0.01	0.28	32.64	15.877	0.486
300	63	0.01	0.30	32.40	16.133	0.498
320	64	0.01	0.32	32.16	16.389	0.510
340	67	0.01	0.34	31.92	17.157	0.538
360	66	0.01	0.36	31.68	16.901	0.534
380	66	0.01	0.38	31.44	16.901	0.538
400	65	0.01	0.40	31.20	16.645	0.534
420	64	0.01	0.42	30.96	16.389	0.529
440	63	0.01	0.44	30.72	16.133	0.525
460	62	0.01	0.46	30.48	15.877	0.521
480	62	0.01	0.48	30.24	15.877	0.525
500	61	0.01	0.50	30.00	15.621	0.521
						0.538

Table A13:_Shear Stress Calculation for OB+30% Flyash with 1.5 kg/cm^2 Load

Sample = OB + 30% flyash

Density (ρ) = 1.70 g/cc
Moisture (M) = 15.95 %

Mould dimensions = 6 x 6 x 2.8 cc
Area of the mould (A0) = 36 cm2
Volume of the mould (V) = 100.8 cc

Amount of sample taken = V x ρ = 171.36 gm
Water added = 15.95% of 171.36 = 27.33 ml

Vertical load = 1.5 kg/cm2
Rate of shearing = 1.25 mm/min

Calibration of proving ring: 199 ↔ 0.50 kN (50.96 kg)

Strain Gauge dial reading (S)	Proving ring reading (P)	LC of Strain gauge (mm)	displacement d = (S*LC)/10 (cm)	Corrected Area A1 = A0(1-d/3) cm2	Applied load L = (P*50.96)/199 kg	Shearing Stress τ = L / A1 kg/cm2
20	26	0.01	0.02	35.76	6.658	0.186
40	35	0.01	0.04	35.52	8.963	0.252
60	39	0.01	0.06	35.28	9.987	0.283
80	45	0.01	0.08	35.04	11.524	0.329
100	48	0.01	0.10	34.80	12.292	0.353
120	52	0.01	0.12	34.56	13.316	0.385
140	60	0.01	0.14	34.32	15.365	0.448
160	68	0.01	0.16	34.08	17.413	0.511
180	74	0.01	0.18	33.84	18.950	0.560
200	81	0.01	0.20	33.60	20.743	0.617
220	87	0.01	0.22	33.36	22.279	0.668
240	92	0.01	0.24	33.12	23.559	0.711
260	97	0.01	0.26	32.88	24.840	0.755
280	101	0.01	0.28	32.64	25.864	0.792
300	103	0.01	0.30	32.40	26.376	0.814
320	104	0.01	0.32	32.16	26.632	0.828
340	105	0.01	0.34	31.92	26.888	0.842
360	106	0.01	0.36	31.68	27.145	0.857
380	106	0.01	0.38	31.44	27.145	0.863
400	105	0.01	0.40	31.20	26.888	0.862
420	104	0.01	0.42	30.96	26.632	0.860
440	104	0.01	0.44	30.72	26.632	0.867
460	103	0.01	0.46	30.48	26.376	0.865
480	102	0.01	0.48	30.24	26.120	0.864
500	101	0.01	0.50	30.00	25.864	0.862
						0.867

Table A14: Shear Stress Calculation for OB+30% Flyash with 2.0 kg/cm^2 Load

Sample = OB + 30% flyash

Density (ρ) = 1.70 g/cc
Moisture (M) = 15.95 %

Mould dimensions = 6 x 6 x 2.8 cc
Area of the mould (A0) = 36 cm2
Volume of the mould (V) = 100.8 cc

Amount of sample taken = V x ρ = 171.36 gm
Water added = 15.95% of 171.36 = 27.33 ml

Vertical load = 2.0 kg/cm2
Rate of shearing = 1.25 mm/min

Calibration of proving ring: 199 ↔ 0.50 kN (50.96 kg)

Strain Gauge dial reading (S)	Proving ring reading (P)	LC of Strain gauge (mm)	displacement d = (S*LC)/10 (cm)	Corrected Area A1 = A0(1-d/3) cm2	Applied load L = (P*50.96)/199 kg	Shearing Stress τ = L / A1 kg/cm2
20	30	0.01	0.02	35.76	7.682	0.215
40	42	0.01	0.04	35.52	10.755	0.303
60	54	0.01	0.06	35.28	13.828	0.392
80	65	0.01	0.08	35.04	16.645	0.475
100	74	0.01	0.10	34.80	18.950	0.545
120	82	0.01	0.12	34.56	20.999	0.608
140	91	0.01	0.14	34.32	23.303	0.679
160	98	0.01	0.16	34.08	25.096	0.736
180	105	0.01	0.18	33.84	26.888	0.795
200	113	0.01	0.20	33.60	28.937	0.861
220	124	0.01	0.22	33.36	31.754	0.952
240	130	0.01	0.24	33.12	33.290	1.005
260	135	0.01	0.26	32.88	34.571	1.051
280	138	0.01	0.28	32.64	35.339	1.083
300	140	0.01	0.30	32.40	35.851	1.107
320	140	0.01	0.32	32.16	35.851	1.115
340	141	0.01	0.34	31.92	36.107	1.131
360	142	0.01	0.36	31.68	36.363	1.148
380	142	0.01	0.38	31.44	36.363	1.157
400	141	0.01	0.40	31.20	36.107	1.157
420	140	0.01	0.42	30.96	35.851	1.158
440	139	0.01	0.44	30.72	35.595	1.159
460	138	0.01	0.46	30.48	35.339	1.159
480	136	0.01	0.48	30.24	34.827	1.152
500	135	0.01	0.50	30.00	34.571	1.152
						1.159

Table A15: Shear Stress Calculation for OB+30% Flyash with 2.5 kg/cm^2 Load

Sample = OB + 30% flyash

Density (ρ) = 1.70 g/cc
Moisture (M) = 15.95 %

Mould dimensions = 6 x 6 x 2.8 cc
Area of the mould (A0) = 36 cm2
Volume of the mould (V) = 100.8 cc

Amount of sample taken = V x ρ = 171.36 gm
Water added = 15.95% of 171.36 = 27.33 ml

Vertical load = 2.5 kg/cm2
Rate of shearing = 1.25 mm/min

Calibration of proving ring: 199 ↔ 0.50 kN (50.96 kg)

Strain Gauge dial reading (S)	Proving ring reading (P)	LC of Strain gauge (mm)	displacement d = (S*LC)/10 (cm)	Corrected Area A1 = A0(1-d/3) cm2	Applied load L = (P*50.96)/199 kg	Shearing Stress τ = L / A1 kg/cm2
20	20	0.01	0.02	35.76	5.203	0.145
40	43	0.01	0.04	35.52	11.186	0.315
60	54	0.01	0.06	35.28	14.047	0.398
80	65	0.01	0.08	35.04	16.909	0.483
100	73	0.01	0.10	34.80	18.990	0.546
120	86	0.01	0.12	34.56	22.371	0.647
140	94	0.01	0.14	34.32	24.452	0.712
160	103	0.01	0.16	34.08	26.794	0.786
180	115	0.01	0.18	33.84	29.915	0.884
200	123	0.01	0.20	33.60	31.996	0.952
220	131	0.01	0.22	33.36	34.077	1.022
240	136	0.01	0.24	33.12	35.378	1.068
260	141	0.01	0.26	32.88	36.679	1.116
280	144	0.01	0.28	32.64	37.459	1.148
300	146	0.01	0.30	32.40	37.979	1.172
320	147	0.01	0.32	32.16	38.240	1.189
340	148	0.01	0.34	31.92	38.500	1.206
360	148	0.01	0.36	31.68	38.500	1.215
380	149	0.01	0.38	31.44	38.760	1.233
400	149	0.01	0.40	31.20	38.760	1.242
420	148	0.01	0.42	30.96	38.500	1.244
440	147	0.01	0.44	30.72	38.240	1.245
460	146	0.01	0.46	30.48	37.979	1.246
480	144	0.01	0.48	30.24	37.459	1.239
500	143	0.01	0.50	30.00	37.199	1.240
						1.246